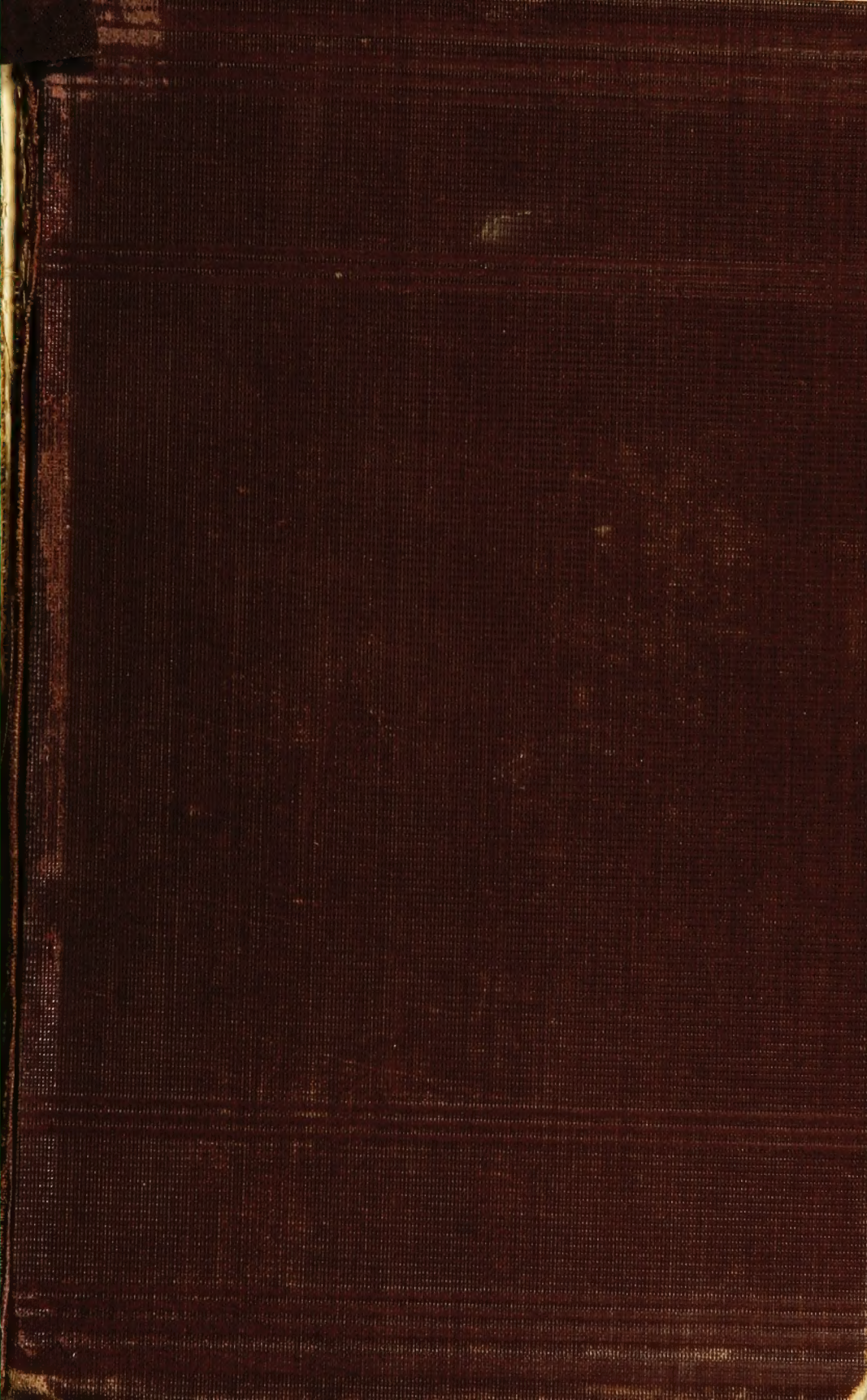




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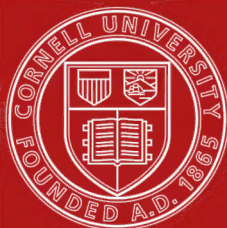
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A
MANUAL OF MINING.

BASED ON THE COURSE OF LECTURES ON MINING
DELIVERED AT THE SCHOOL OF MINES
OF THE STATE OF COLORADO.

BY

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THIRD REVISED AND ENLARGED EDITION.

FIRST THOUSAND.

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TO
Ogden H. Rood, A.M.,
PROFESSOR OF PHYSICS, COLUMBIA COLLEGE, NEW YORK CITY,
WITH RESPECT AND ADMIRATION, THIS
VOLUME IS DEDICATED
BY
THE AUTHOR.

PREFACE.

THIS treatise is an abbreviation of a course of lectures upon mining, delivered at the School of Mines of the State of Colorado, and is issued with the advice of its Board of Trustees, which recognizes the importance of having, within a moderate compass, the best information obtainable upon this subject. In its presentation, the writer has followed what his own experience has taught him to be the natural sequence, and has endeavored to introduce such matter as sixteen years of lecturing and field work have suggested as requisite. Part I contains a brief geological review and a discussion of such points as the engineer must include in his report, i.e., the preparatory and development work, systems of mining and the plant for power, hoisting, pumping, and ventilation. Part II embraces the practice of prospecting, drilling, blasting, shafting, tunnelling, and timbering, in addition to some remarks upon the examination of mines.

The work is designed as an elementary treatise for the use of those desiring a reference-book. The complexity of the subject, its extent, and the variety of machines to be described and represented, demand an elaborate discussion that would fill several quartos. Descriptions of obsolete and expensive systems or machinery are relegated to the historical works on mining. American and foreign practice is described; and suggestions for lines of future progress are offered herein. The principles of the construction and operation of machines used in mining are explained with a perspicuity and conciseness compatible with the field in which this publication is to be sown—among students and mining men, to whom a knowledge

of the fundamenta of their work is valuable, but whose acquaintance with the theory is slight.

The wants of the latter class have been kept in mind, and the writer hopes that the manual may prove of some benefit to the intelligent reader, of whom it presupposes an elementary knowledge of the sciences and of the simple machines.

The author regrets his inability to deal with the subject of "electricity in mining" as it deserves. Two reasons account for this: insufficient data, as yet; and the large space which a satisfactory explanation of the principles would demand.

The writer would also beg leave to say that the literature of mining and its cognate branches has supplied much of the material contained herein. References could not be made for each hint obtained, but obligations are acknowledged to the authors of the publications mentioned, to which the reader is referred for further details. The information has been garnered from the best available sources and condensed. The *Engineering and Mining Journal*, the *Colliery Engineer*, and the transactions of the *American Institute of Mining Engineers* have been copiously drawn from, as also the experience of the practical men, to a long list of whom the Author is indebted for many courtesies. Finally, to the manufacturers and engineers thanks are rendered for the use of the electrotypes, which have so largely contributed to make the work attractive.

MAGNUS C. IHLSENG.

GOLDEN, COLORADO, Nov., 1891.

PREFACE TO THE THIRD EDITION.

IN the present issue the text has been increased by more than fifty pages in order to introduce additional matter pertaining to the design of cars, hoisting appliances, and fans, the added illustrations having been specially prepared with that end in view. At the end of each chapter will be found a list of memoirs which have been carefully selected to comprehend the latest literature on the subject. The original intent to also enter at some length upon a discussion of the applications of electricity to mining was abandoned after a number of fruitless efforts to compress the matter into the small compass which it should occupy in a work of this nature.

It is gratifying to observe the progress made in the increased economy in the utilization of power. The growth of the young giant among the motor fluids in the anthracite regions is particularly noticeable; while that of electricity in its various applications in bituminous mines is rapid and sure. It may be true that the adoption of these motor agents by mining engineers has been a little slow, but this wise conservatism is dictated by the knowledge that a single accident resulting from the introduction of an innovation might precipitate a peremptory legislative prohibition which would render the outlay entirely useless, and it is with reluctance that they must frequently forego the advantages of some possible economic installation.

It is gratifying to observe the extended employment of

artificial methods of ventilation in the metalliferous mines of the West, and when the American method of "square sets" shall have been supplanted entirely by the method of rock-filling, or of flushing with waste, the dangers to life and property will have been further reduced and the economy of mining materially increased.

Regarding the enlargement of the chapter on explosives and the stress laid upon the necessity for an absolute prohibition of black powder from the coal mines, I hope they may be fruitful of results.

Finally, there has been added at the end of each chapter a list of the more important memoirs dealing with the subject matter of that chapter, and this, carried to the close of the year 1897, has brought the work up to date. It should prove of good service as a syllabus.

The author hopes that the book in its revised form will contribute to a better circulation of the knowledge of the principles upon which mining engineering is founded.

MAGNUS C. IHLENG.

STATE COLLEGE, PA., Dec. 20, 1897.

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 Norwalk Iron-Works, South Norwalk, Conn. 258, 262.
 Oil-Well Supply Co., Pittsburg, Pa. 212, 213.
 Pelton Water-Wheel Co., 123 Main Street, San Francisco. 33.
 H. K. Porter & Co., Pittsburg, Pa. 66, 67.
 Rand Drill Co., 23 Park Place, N. Y. City. 3*, 5*, 115, 233, 234, 240, 252, 260, 261*.
 Spiral Weld Tube Co., 43 John Street, N. Y. City. 82, 83.
 William E. Stieren, 544 Smithfield Street, Pittsburg. 98, 106-109.
 Trenton Iron-Works, Trenton, N. J. 71, 72, 74.
 Webster, Camp & Lane Co., Akron, Ohio. 29*.
 Yale & Towne Manufacturing Co., Stamford, Conn. 30.

ABBREVIATIONS.

THE following is a list of abbreviations which are used to denote the periodicals to which references may be made for a more extended discussion of the subject than is given in the text. The title of the article, the name of the author, the volume or date of the publication, and the page on which it is found are given in the order named.

- A. M. Inst. M. E.* Transactions of American Institute of Mining Engineers, 19 Burling Slip, N. Y. C.
- Am. Mfr.* The American Manufacturer and Iron World, Pittsburg, Pa.
- Ann. des Mines.* Annales des Mines, St. Etienne.
- Brit. Soc. Min. Stud.* British Society of Mining Students, Radstock Colliery, Bath, Eng.
- Bureau of Mines, Ontario.* Reports; Toronto, Ontario.
- Cal. State Min. Bureau.* Report of California State Mineralogist, Sacramento, Cal.
- Cassier's Magazine.* The Cassier's Magazine, World Building, N. Y. C.
- Chest. Inst.* Chesterfield and Midland Counties Institution of Engineers, 15 Cavendish St., Chesterfield.
- Coll. Eng.* The Colliery Engineer, Scranton, Pa.
- Coll. Guard.* The Colliery Guardian, Strand, London, W. C.
- Coll. Mgr.* The Colliery Manager, Bowerie St., London, E. C.
- Elec. Eng.* The Electrical Engineer, 208 Broadway, N. Y. C.
- Elec. World.* The Electrical World, 253 Broadway, N. Y. C.
- E. M. J.* The Engineering and Mining Journal, 253 Broadway, N. Y. C.
- Eng. Asso. of the South.* The Engineers' Association of the South, Birmingham, Ala.
- Eng. Magazine.* The Engineering Magazine, 120 Liberty St., N. Y.
- Eng. News.* Engineering News, Morse Building, N. Y.
- Eng. Rec.* The Engineering Record, 100 William St., N. Y.
- Eng. Soc. W. Pa.* Proceedings of the Engineers' Society of Western Pa., Pittsburg, Pa.
- Fed. Inst. M. E.* Transactions of the Federal Institution of Mining Engineers, Neville Hall, Newcastle-upon-Tyne, England.
- Frank. Inst. Jour.* Journal of the Franklin Institute, Philadelphia, Pa.

- Geology Survey Ohio.* Columbus, Ohio.
- Ill. Min. Inst.* Transactions of Illinois Mining Institute, Springfield, Illinois.
- Jour. Asso. Eng. Soc.* Journal of Association of Engineering Societies, Chicago, Ill.
- L. S. Min. Inst.* Lake Superior Mining Institute, Minneapolis, Minn.
- Min. Bureau, Col.* Bureau of Mines of Colorado, Denver, Colo.
- Mineral Industry.* Statistical Volumes. Scientific Publishing Co., 253 Broadway, N. Y. C.
- Mines and Minerals,* Scranton, Pa.
- Min. and Sci. Press.* The Mining and Scientific Press, San Francisco, Cal.
- Min. Bull.* The Mining Bulletin of the Pennsylvania State College.
- Min. Inst. Ill.* Transactions of the Illinois Mining Institute.
- Min. Ind.* The Mining Industry and Tradesman, Denver, Colorado.
- Mine Inspec.* Reports of Mining Inspectors of the State named.
- M. & M. Eng. Trans.* Transactions of North England Mining and Mechanical Engineers, Newcastle upon-Tyne, England.
- N. Staff. Inst.* Proceedings of North Staffordshire Institute of Mining and Mechanical Engineers.
- N. E. I.* Transactions of North of England Mining and Mechanical Engineers.
- Ohio Min. Jour.* Journal of Ohio Mining Institute, Columbus, Ohio.
- Queensland.* Report of Secretary for Mines, Queensland.
- Rev. Univ.* Revue Universelle des Mines, Liège.
- S. of M. Quart.* The School of Mines Quarterly, Columbia University, N. Y. C.
- Second Geolog. Surv. Pa.* Report of Second Geological Survey of Pennsylvania, Harrisburg, Pa.
- Scientif. Quart.* The Scientific Quarterly of the Colorado School of Mines, Golden, Colorado.

MANUAL OF MINING.

PART I.

MINING ENGINEERING.

CHAPTER I.

GEOGNOSY.

1. Bird's-eye view of the subject ; native metals, minerals, ores, and their occurrences, definitions. 2. Vein matter, gangue, and gouge ; geognosy of ore-deposits ; gash and fissure veins, beds and blankets ; geological theories and miners' rules ; prejudices and fallacies regarding ore-deposits. 3. Prospecting ; searching for veins ; indications, float, shode, and slide rock ; examining new districts ; divining-rods, spiritual mediums, and the drill as miners. 4. Remarks upon the chaotic state of the U. S. mining laws ; apex *vs.* side lines ; safety in the side-line law ; advice to locators ; insecurity of locations on the apex ; patenting claims.

1. The search for the useful and precious minerals has been diligently prosecuted since the early days of civilization ; their discovery and application have made nations powerful exponents in the world's history. And nowhere is this fact better exemplified than in our own land, in the wonderful opening and rapid settlement of the Western mining States.

No subject is more entrancing, no occupation more exhilarating, than mining, with its wonderful kaleidoscopic changes. In early times excavations were made and mines worked only to a small depth and in easy rock, and that, too, only for sub-

stances of high intrinsic value, notwithstanding the myriads of slaves to furnish the labor. The attempts at systematic mining were few and far between; but since the advent of the steam-engine, mining has been acknowledged an important profession, requiring technical education. Competition with the whole world, brought about by the improved means of communication, the paucity of bonanzas and their rapid exhaustion, compel a skilful utilization of all the aids to a cheap extraction of our immense wealth.

The accessibility of the mine and the vendibility of its product are the ever-ameliorating features in the mining history of nations, districts, camps, and individuals, gradually divesting mining of its risks and rendering it more and more akin to manufacturing. Each new camp, untrammelled by tradition to keep it in the rut of prejudice, displays its genius for organization and absorbs the latest devices, tried and true. Nevertheless, it must be admitted that in each camp an adequate solution of the problem involves intricate questions of environment. The economy of mining is a function of many variables, as geological stratigraphy, subterraneous uncertainties, wages, water, timber, transportation, and treatment. The constants are few. The proper relation of these it is our province herein to discuss.

Hitherto a gambling spirit has frequently controlled investments in metal mines. Speculative tendencies, not technical economies, have dominated some of our operators; their heavy aggregate outlay may have proven unprofitable, for the present, because of salted mines, attractive prospectuses, or incompetent management. It must be remembered, however, that they have contributed to the prosperity of the country, and at some later date their abandoned exploitations will be pursued to profit, when the potential investment of to-day will have been resolved into future kinetic dividends, the cost of production being continually on the decrease.

The occurrence of the useful or precious minerals in the state of native purity is rare. Still less often are they found superficially: they must be delved for. In the extraction of

this subterraneous material, and its delivery to the surface, consists the art of mining. The legal definition of a mine includes such "workings as must be artificially lighted."

Gold and platinum are found native in the placer accumulations of ancient and modern river-beds, which furnish fully 75 per cent of the total output of these metals. Gold occurs in segregated veins, alloyed with tellurium, and always associated with pyrites and titaniferous iron; also intercalated between the sheets of slate or shale, or finely disseminated in eruptive rocks. The only extensive native copper deposit is the remarkable product of the Lake Superior region, where the irregular masses are mined out of the amygdaloid trap and sandstone. Singular masses of metallic iron ore are found in several localities, but they are curiosities and casual, if not meteoric. Native silver is rare and occurs in Peru, Mexico, Norway, and in the Lake Superior copper mines.

With these few exceptions the metals are encountered in chemical union with non-metallic substances, more or less completely segregated to constitute mineral. Any accumulation of mineral of good quality and in sufficient concentration to warrant the expenditure of energy for its extraction is an ore. Manifestly this is a fickle term, since it depends for its stability upon the casual conditions of the market as well as upon the mineralogical features.

The most common substance is iron, entering as it does into almost all rocks and veins. Its most frequent, and valueless, combination is with sulphur. Magnetic and specular oxide and the carbonate constitute the entire supply. These occur as irregular masses in the rocks of every geological age, or in veins mixed with other minerals, but are chiefly in the metamorphic crystalline, Archæan rocks. Zinc is obtained from calamine, franklinite, and blende, which are quite extensively distributed in the Carboniferous strata. With very few exceptions, galena is exclusively the ore of lead. The carbonate and the sulphide, in the lower Silurian and Carboniferous strata, mostly occur in irregular shoots and pockets, and rarely argentiferous. In the older metamorphic rocks the galena is con-

finer in fissure veins carrying silver and gold. The main supply of silver is from its minerals, more or less intimately associated with other ores. Similarly with them, it has a wide geological distribution, and is also found "dry" in fissures. Copper, as chalcopyrite, bornite, and cuprite, is disseminated in and along slates and sandstones, rarely above the Triassic. Many galena veins in the metamorphic rocks change with depth to copper. Mercury comes from cinnabar, which is found in true veins and in contacts. It is not commonly encountered. Tin has a characteristic occurrence in but one form, as an oxide, and only in gash or segregated veins, or "stockwerke" of the older rocks.

Tin lodes are of the segregated type, and gold or silver bearing, pyrites and cassiterite being the common minerals.

Millerite and pyrrhotite are nickeliferous and occur in gash and segregated veins, rarely deeper than 500 feet. Rich films of genthite in talc veins often constitute a commercial supply.

Manganese ores (standard contains 44 per cent of the metal) are generally associated with limonite and occur in pockets usually embedded in clay as contacts or beds or permeating slates. Films of manganese appearing in moss-like forms on the face of rock give it the name of "landscape" rock.

Mica is generally in bedded veins, instances of contacts and true lodes being rare. They are simply and always dikes in coarse granite. Hitherto only large slabs were sought, but now the fine, clean mica has a ready sale for lubrication and other purposes.

Phosphate rocks for fertilizers, the practical value of which is determined by the amount of phosphoric acid contained, are found as beds of irregular thickness; veins or lodes transversely to the strike of the strata; or superficial deposits. Apatite occurs concretionary in a clay matrix between limestone and clay. These are more frequent in the Miocene.

Many of the metals are incidentally obtained from their mineral compounds while smelting for other metals with which they are associated.

The metalliferous portion of a lode usually comprises only

a small portion of its contents. The argentiferous galena, bornite, blende, or their oxidized derivatives in grains, pockets, or streaks, more or less connected, are associated with a "gangue" of clay, quartz, fluor, calc, or heavy spar. These earthy materials sometimes are intimately mixed with the mineral, and again lie in layers contiguous with it, or the different constituents may even manifest a ribbon-banded structure.

The entire mass, metalliferous and earthy, constitutes a deposit which is known as a bed or a vein, and may exist under such circumstances as to render it workable. The term *vein* is intended to describe a regular unstratified deposit in a fissure that traverses the country for a considerable distance, longitudinally and vertically. The Supreme Court has defined it as "any zone or belt of mineralized rock lying within boundaries clearly separating it from the surrounding rock." This demands a well-defined crevice of ready identification, and two solid walls to give it individuality. Its lead must be metalliferous. A vein is the filling of a pre-existing fissure. The term has lost the significance it once had. The mineral system was originally supposed to have a resemblance to the human circulatory system. True, the fissures have originated during periods of great dynamic movement, producing folds and fissures which are supposed to have extended deep into the earth's crust, but the main artery has yet to be located. Though argentiferous lead veins are quite persistent, no evidence exists for the dogma, so tenaciously held, that they increase in richness with depth. They may or may not become richer, or change, in constituents. Examples can be cited for either side of the argument. In folded strata the deposit inclines to be thicker at the ridges, or troughs, and thinner at the sides of the folds. But this is not generally the case in massive rocks.

Usually the vein matter is crystalline. It is commonly separated on either or both walls from the surrounding rock by a sheet of clay (called "selvage" or "gouge"), or by other quite distinct lines of demarcation. The surface of contact of the deposit with the adjacent rock is called a wall, roof, or

floor, according to its relative position to the miner. Not infrequently the walls are polished surfaces ("slickensides"), due to grinding caused by the slips during nature's contortions. Sometimes portions of the vein have slid on one another, causing "false walls"; therefore the miner is advised to occasionally break into the walls to assure himself as to the fact. On the other hand, a vein may have only one or even no wall. In the process of mineralization, the original face or faces of the fissure may have become disintegrated, and all evidences of the looked-for wall obliterated. In such cases, economic, not geologic, or legal conditions define the vein.

2. Fissures belong to regions of metamorphic action, and are the principal repositories of the precious metals. And it is a striking fact that they are rarely found singly, rather in groups of parallel veins, often in congeries. *Stockwerke* is a term used to describe a condition of affairs in which the country rock is creviced in all directions, so that the whole mass must be mined out. Some are filled with eruptive matter, others with vein matter, still others were subsequently closed without any deposition. The mineral components are markedly dissimilar, and indicate different sources. Those filled with the same variety of mineral were doubtless produced by contemporaneous forces. Those fissures which interrupt the continuity of the older veins are called cross-courses. The manner in which the intersections occur determines their relative age. Their absolute age is not ascertained, unless in stratified rock. Drags are more common than is supposed, and should not be confused with intersections. The latter are usually richer, the former not necessarily so, at the point of juncture. Many of the older veins are broken and displaced by faults. Not only do veins "pinch and shoot," but the pay streak will vary in thickness, plunge from wall to wall, or split up into numerous feeders and ramifications, and even disappear in a thread.

Gash veins hold a subordinate position to fissures. But they are of small extent, and are usually confined to a single member of the formation in which they occur. Their habitat

is unmetamorphosed sedimentary rock. They have no distinct walls or gouge, and are unreliable.

The most important sources of the mineral wealth are the metalliferous deposits which occur in the sedimentary strata, and are termed beds. While the geologists may classify them, the group is sufficiently identified by this term for mining purposes. It includes deposits, somewhat irregular in dimensions, occurring in the transverse joints of the rocks; as cementing material to the remnants of shattered or insoluble rock; as layers conformable with the strata; as isolated impregnations of grains or bunches in porous rock; or as a metasomatic replacement of porous rock. They may be found similar to fissures in a certain formation, then as a blanket contact parallel to the stratification, to again plunge into a lower series of rocks like a fissure, or branch out into a chamber. They are more easily mined, but are less persistent in depth, than veins. Their mineral contents are very compact, seldom crystalline, and the gangue hardly distinguishable from the country rock. The mineral is more or less concentrated along certain lines called "ore-shoots," which probably constituted the channels of communication with the ultimate source. The same is also true of veins.

Irrespective of any theory, one requisite condition for deposition is a crevice, a porous or soluble rock conduit for the fluid from which local action has precipitated the mineral. Open cavities were not necessarily pre-existing, for a vesicular rock would allow of an easy flow to the magma, or it might be equally well secured by dissolving action on the rock and a subsequent replacement. This is independent of its geologic position. In every age are rocks which will satisfy this condition. Besides this, a long train of circumstances has preceded the vein-formation involving dynamic agencies, heat and metamorphism, and even eruptive action, as important factors. These disturbances having been often repeated through the different ages, the older rocks were more frequently shaken up. Beyond this no reason exists for the prejudice which favors certain geological formations as ore-bearing.

The geognostical relations between veins and their contents are of importance to the mining engineer, but our limited space will not admit of any discussion here. The various works on geology will supply the information as to the vagaries manifested by ore occurrences and the numerous theories held. Some isolated examples exist under such circumstances as to suggest the same origin for the ore as for the adjoining rock-formations. Many of the beds and veins have been impregnated by percolating waters, perhaps at high pressure and temperature, contemporaneously with the country rock. Their metallic contents may have been carried in solution or they may have been in a molten or a gaseous state when the way for their passage was opened.

This is a matter for conjecture, as is also the ultimate source of the mineral. Certainly the evidences point to its deposition as a sulphide, the oxidized forms being accounted for by long-continued action of atmospheric agencies. The presence of coal and bitumen in many lead and zinc veins and beds in a measure suggests a theory of cause. The "water-line" theory has served its day and is no longer tenable. The current theories have, for hundreds of years, afforded satisfactory explanation of the genesis of some of our ore-deposits. But when we find contiguous depositions contrasting widely in point of density; narrower parts of fissures filled by larger deposits or richer ores; superior minerals higher up than the more volatile or lighter ones, even alternating with them, we must admit that since the day Job declared that "silver is in veins," little material progress has been made by our geologists beyond the slow garnering of facts which, ultimately, are hypothesized. Our knowledge upon this branch is cumulative and in expression conservative. However, any theory explains some, but none all, of the capricious examples of lodes or their anomalous fillings. The veins we find, but not always the silver; and this inability to formulate a general law by which to locate the hidden bonanzas has led to the compounding of the numerous witcheries, and divining-rods of every conceivable form, for imposing upon the credulity of the prospector who

seeks a quicker means of acquirement than is afforded by the use of the pick, shovel, and patience.

There is no particular angle of dip or bearing of trend that is universally favorable to rich veins. Rules based upon such observations are local only. The same may be said as to the supposed "live"-ness of certain rocks to mineral. Attempts to formulate indications of "quickenings" mineral by associations with certain gangue matter or minerals have failed of generalization. The mineral is where you find it. The Cornishman's adage, "riding a zinc horse to fortune," has no verity in this country. Each locality has its own peculiarities of mineralization, which the careful and systematic engineer will observe and regard.

3. With the two classes of rocks, stratified and massive, are coexistent the two classes of mineral deposits, beds and veins. Though many occurrences are of a nature that admits of question as to classification, for mining purposes a sharp line of distinction is not sought. Legal technicalities have so confused the definitions of deposits and veins as to obliterate all semblance to the original intent of geologists and mining men. Of this, more later. At present we shall consider some rules to assist the prospector in his search for mineral. And while it must be admitted that many a find has been made through accident, the existence of the ore would be found not to be at variance with the cumulative rules of geologic science.

Accordingly, the prospector will seek within geological confines. In regions of stratified rock the matter is simple. Coal is found in three geological horizons, and the presence or absence of the rocks belonging thereto is indicative of the prospects.

The metals and their minerals are distributed, geologically and geographically, over a large extent. The zinc ores in this country occur in the Carboniferous and along the Mississippi valley. The Archæan and Silurian are most prolific of the other ores. The precious metals are chiefly found in the mountainous districts, because the phenomena attendant upon

their formation were conducive to the filling of veins, and the forces which gave character to the mountain also impressed themselves upon the vein, which is exposed to view and subject to location. Without some such providential occurrences to change the monotonous topography of the preadamite surface, bedded veins of the stratified districts would have been revealed only by boring, while those in massive rocks might never have been formed.

Surface prospecting is confined, therefore, to the seeking for an outcrop. In igneous rock the outcrop is easily found. For, unless the hill is covered with slide rock, it is indicated by a jutting ledge (if the vein matter is harder than the country rock), or by a sag (if it is decomposable). In heavy timber this may go unnoticed. At high altitudes snow in the sags calls attention to the leads.

The same is true of coal, which is located by the terraces which mark the outcrop. The trend of the terrace, relative to the topography of the hill, gives a good idea of the slope of the coal. The bench itself may give the desired information, but usually it will be found that the coal dips with the hill, when the terrace or depression deflects outward toward the bottom of the hill, and the reverse for a coal dipping inward, when the outcrop will be concaved toward its top.

Substances foreign to the rock deserve notice. Alternations in the color of the slide rock covering the hill are good indications of the presence of oxidizable minerals above. So, too, vegetation is a guide. Iron springs often accompany the outcrop of coal; the ochreous covering of the rocks and soil is noticeable near some of the anthracite seams, and is common in the semi-bituminous districts. Masses of highly oxidized matter, broken from the veins, compose what are called "blow-outs," and are common in galena regions.

If no evidences of outcrop are thus found, "booming" may disclose it. During winter or a wet season, snow or water is collected in a reservoir upon the hill, and, at a convenient time, turned loose to plough its way over the soil in its fall. Many a vein has been thus discovered without great expense.

In stratified regions the order of the geological series may be observed, and certain fossils furnish the guide. Or, if the prospector is examining new ground, he has but to look for mineral in the float on the surface or in creek-bed. The appearance of material derived from erosion is indicative of the character of the rock from regions higher up. Therefore the bed of the stream, or the hill slope, is minutely examined for fragments of ore, or blossom, and followed as long as mineral is found. If the float or shode boulders are pebbly or rounded, or in vegetable soil, they have come from afar and the lode is not at hand. If the shode is large and angular, it has not come very far, and the discovery of a point beyond which no float or blossom is detected is presumptive evidence of approach to the vein. The lode will be found above the point of discovery, and the prospector will go in the direction of the drainage and thoroughly search the ground.

In high altitudes the oxidation of the minerals in, and the electric manifestations of, the vein outcrops have assisted the prospector by the light playing over them. This is of continued occurrence in Colorado above timber line, and particularly in regions of arsenical veins.

When found, the vein should be examined, and its value confirmed at several points ; most monstrous disappointments have ensued from testing of the lode at one point only. If the country is stratified, care is taken to ascertain all the data of thickness, etc. Frequently the ore oxidizes and rots away, to be crushed by the overlying strata showing only in a small streak. Or the outcrop may fold back, "tail out," and give false impressions of great thickness.

Maps are serviceable as showing the important features, and a systematic plotting of all data, geological and otherwise, gives a good basis for conclusions. Dr. H. M. Chance, in the Second Geological Survey of Pennsylvania, has an admirable discussion on the construction of geological cross-sections, to which the reader is referred. Prospecting for oil or gas is speculative, and the sole guide is the geologist's facts. Reports I, and J

of the Geological Survey of Pennsylvania, the Treatise on Petroleum by Benj. J. Crew, and the report by S. F. Peckham to the Census Bureau are monographs on the subject.

If surface examination fail to give a trace of mineral sought, and there remains reasonable expectation of finding it, a tunnel, a shaft or boring may be resorted to. The two former are more expensive but safer guides than that offered by boring. Shafting is slower and more costly than tunneling, but more quickly reaches a flat seam at a point suitable for development. The steep pitching vein is perhaps best reached by a tunnel, if the depth of vein so gained is great enough to compensate for the length of tunnel. The choice between them depends upon local conditions. Both are advisable for shallow explorations, while drilling may be employed for deep work. The latter is very commonly employed on account of its cheapness. But even when it has determined the data, previously doubtful, the shaft or tunnel has to be subsequently driven. So drilling has its limitation of use. It is rarely employed as a seeker for mineral, but merely to give confirmation to, and assist in a rational estimate of, the value of the undertaking. Many properties owe their rehabilitation to the results of the diamond-drill exploitation, and none should be abandoned until after a careful surface examination had been made and followed by numerous boreholes.

Either the punch (72) or the diamond-drill (88) method may be used for the boring. The former is cheaper, but the pulverulent material brought up by the sludger is unsatisfactory; it may indicate the constituents of the rocks pierced at different depths, but can give little of its physical character or dip. The diamond-drill core gives a little more information, but even its indications are hardly trustworthy. It affords an opportunity to identify the rock, but some of the soft strata is worn away and the core may be turned in its tube, so its revelations are not much better than those of the sand of the punch-drill, which is faster. The smaller diameter of the

hole of either renders its results doubtful; for it may have just missed the mineral, or have struck a solitary, small, soft chunk of ore, which would supply cuttings to discolor the sands for a long distance and give amazing report. Very important deductions cannot be based solely upon the indications of the borings. Only after numerous holes and a satisfactory surface examination can a conclusion be reached.

Good, hard common-sense, observation and pluck win, and they alone. There is no mystery about the finding of mineral. Nature is bountifully supplied with precious metals and valuable minerals, but her secrets are hid. Only the cumulative information of geological experience gives any clue as to the habitat. Neither witchery nor magic charm can hasten the knowledge of the whereabouts of an ore body or deposit.

The wizard with the hazel wand, or the spirit medium who is controlled by some disembodied Comanche chief, is an impostor. No sooner is he thus equipped than he affects a versatility and occult power that transcends combined scientific knowledge. Nevertheless, to a paltry amount of "filthy lucre" he is not averse, when he plays upon the credulity of natures which are duped to making extensive explorations upon the purported previsions. This would be ludicrous, were it not also painful, to see the number of misguided men who have squandered hopes and possessions in their search for a short-cut to wealth.

4. The discovery of mineral at the surface must be followed up to prove the existence of a lode or vein. The existence of an ore-deposit is a stratigraphical fact which is demonstrable, and the granting of mining rights under the law is accorded upon this proof. The General Land Office of the United States and the courts decreed that a single shaft does not necessarily carry evidence upon this point. Besides the exposure of an outcrop or an apex on the surface, the existence of a mineralized vein, or of rock in place underneath, is an essential feature. If the ore-body underneath is not a vein, then the concurrence of mineral at the surface is not a part of

a vein. The vein may have become disintegrated; but if the general features still prevail—a crevice carrying mineral matter between rock of a nature and origin different from it—a valid location may be made thereon. If, however, the vein matter has been transported by the elements and become mingled with other rock, it has lost all identity with its lode.

The number of legal definitions of veins is equal to the number of judges who have passed upon the cases. But as the U. S. statutes divide mineral ground into veins and placers only, the presumption would be that any well-defined metalliferous crevice, capable of ready identification by the miner, is a vein, whether fissure or not,—only it cannot be a placer.

The difference in the grants under the two cases, besides a question of acreage, is that the mining of ore within placer ground is confined to the vertical planes through the boundaries (sec. 2329, U. S. Rev. Statutes), while vein deposits may be pursued along their dip, “throughout the entire depth,” even if they “so far depart from the perpendicular” “as to extend outside of the vertical side lines of the claim;” and the extent of the miner’s right is determined only by the vertical planes through the end lines, which should therefore be properly drawn.

Locations 1500 feet in length are permitted upon the public domain to the discoverer of the lode. But for access thereto, and for convenience of working, the U. S. grants, as incident to the principal feature, surface ground which, measured from the middle of the vein, shall not exceed 300 feet on either side. Some States have reduced this to 150 feet on each side, while in some Colorado counties only 25 feet was, and 75 is, the outside limit. The claim must be essentially a parallelogram. It may be 1500 feet, or less, in length, located substantially along the middle of the apex, across which are drawn two parallel end lines and side boundaries, within the limit prescribed, parallel in pairs following the contortions of the outcrop. However else the Act may be vague, it certainly is not upon the fact of the parallelism of the exterior boundaries. Excessive locations are valid as to the legal limit and void as to the excess.

It is incumbent upon the locator to define the boundaries of his claim, by placing stakes at all corners and intersections, to notify others that the ground is entered upon and being exploited. These, with the filing of a location certificate in the county, maintain possessory right from the moment of posting a location notice of discovery upon the lode. Within a reasonable time thereafter, sixty days usually, the locator is required to sink a "discovery" shaft at least 10 feet into the vein. This satisfies the regulations regarding discovery, and maintains a mining right against all comers until the expiration of the calendar year.

From that time on, an "assessment" of \$100 must be expended annually as evidence of mining intent. A failure to expend such sum constitutes a forfeiture by which the claim reverts to the public domain, and is subject to relocation. As the value of assessment work is a matter of opinion and not easily proven, it is safer to each year file an "affidavit of labor," certifying to the assessment work for that year having been performed.

A prospector is not confined to a single entry upon a discovered lode. He may appropriate as many claims as he chooses, contiguous or otherwise, with that of the first discovery. Upon each 1500 feet, or less, of length he must show the intent to mine, by a discovery shaft and the assessment work.

For the development of the mine, the annual assessment work may be done upon the surface or upon the vein, and all efforts outside of the limits of the location with a bona-fide intent to work the claim are justly considered as if upon the claim—as, for instance, development by tunnel instead of shaft.

This concession is further extended by the U. S. Supreme Court; for where one person owns several contiguous claims, capable of being advantageously worked together, one general system of development may be adopted, after the discovery shafts are driven. This encourages more economic work and subserves the best interests of all concerned.

The principle having been fixed, it is not remarkable that

further concession was granted. "Where many claims are consolidated in the hands of one company, there is no impropriety in calling it one mining claim." This rule, adopted by the U. S. General Land Office, solves many harassing questions, but is more prodigal with the public mineral lands than was contemplated by the framers of the mining code.

When, therefore, a vein or rock in place is discovered on the public domain, it may be located on and operated. When the locator has demonstrated his ability to develop the mineral resources of his claim by the expenditure of at least \$500, he may proceed to the purchasing of the land from the United States, i.e., "patenting" his claim. Certain preliminaries are necessary: a survey approved by the U. S. Surveyor-General for the State; notification to the public, by descriptive notices posted on the claim, in the U. S. Land Office, and published in the nearest newspaper for a period of sixty days; affidavits of citizenship and of the execution of the preliminaries; abstracts of title, and the payment for the land at \$5 per acre or fraction.

An exclusive right of enjoyment "of all veins" cropping inside of the boundaries is given with the claim. These ancillary veins and their contents, to any depth whatsoever, cannot become the property of another, even if they are discovered and entered upon in adjacent territory. The subsequent locator, according to the laws of the State of Colorado, may have right of way through the cross-vein to his ground on the other side of the prior claim, but none of the mineral. In every case it is intended that priority shall govern. Sec. 2326 grants to the senior locator the mineral at the intersection, and to the junior the right of way through it.

By the interpretation of the U. S. Statutes, easement and title were clearly intended to be conveyed for all forms of metalliferous deposits, in the use of the terms "veins, lodes, or rock in place." The Act recognizes any mineralized rock in place, enclosed in the general mass of the mountain, as a vein. The arbitrary classification by geologists into veins, beds, and irregular deposits is unimportant in relation to this matter. Whatever the theory of vein-formation may be, it is positive

that crevices were formed during certain convulsions of nature ; in these ore-deposition may have occurred simultaneously with, or subsequently to, the fissuring, giving rise to various forms of veins. The dynamic disturbance and the atmospheric agencies that followed still further modified the geological and topographical features of the country. The processes were more or less similar, but the results are distinguished by geologists by the terms of beds, blankets, fissures, veins of impregnation, of infiltration and contacts. The legal expert has confused these terms.

The Statutes favored the miner and assumed to cover all lodes whose indications were sufficiently marked for the miner to continue explorations thereon. A crevice, crevice matter, a fair wall, and mineral are the essential conditions.

The discovery in Leadville of an outcropping bed, to which a few lucky prospectors were entitled, was followed by the promulgation of a " side-line " theory, the common law of Leadville, Colo.

It has been seen that a lode claim, whether patented or not, carries with it all that is beneath the surface-ground claimed, with a servitude upon the adjoining territory obtaining the right of following the dip of the vein, and subject to a like easement granted to the locator on adjacent ground to pursue his vein wherever it may go. This obtains until some one can show a better right. The common law as to realty is modified when applied to mining property.

It has, however, happened that rulings were so made and construed that a party may locate vacant ground and maintain ownership to the mineral covered by it, unless *it is shown that the mineral body belongs to a lode cropping elsewhere within legally claimed ground*. The proprietor who calmly continued work upon his discovery found himself breaking into the subterraneous workings of others who had stolen a march on him. To secure his right he must bring action to eject. To vindicate his title he must prove the lode is in place and continuous from the point of his discovery *to, into, and through* the

ground of the trespasser. Failing to do which, his claim is defeated and all incidents thereto attached fail.

Naturally the train of reasoning led farther and farther away from the original intent of the law to reward the discoverer of an apex, until the accepted idea is that, although the "defendant's location may appear to you to be along the line of the top, apex, or outcrop of the vein, it cannot prevail against a senior location on the dip" of the lode.

Again, Judge Hallet makes this observation, which, unfortunately, has not been passed upon by the Supreme Court :

"I will say to the counsel in that case [a location made on the middle part of a lode, or otherwise than at the top or apex], which is not for the consideration of the jury, that it has always been a question in my mind whether a location on the dip of a vein would not be valid as against one of later date higher up. That is to say, whether if a location be made upon the dip of a vein, the locator may not pursue it in a downward course, although he may not in the upward course, and may not hold the whole which lies within his location and below it, as against any one locating subsequently at a higher point on the same vein."

A lack of development at the time of hearing in court, a lack of other proof of the "perfect continuity" of the vein from apex to side line, or one imperfect wall, invalidates the title of apex claimant to a lode claim, and the deposit is condemned to be a placer, on the doctrine that the law recognizes no presumption in favor of the existence of a vein, but treats each local aggregation of ore as a separate lode. The decree of "no lode" cuts off the privilege, nay, right, to pursuit along the dip, and permits extraction only within the boundaries of the claim.

To what absurdities the law has led us, by reason of the vagarious interpretations, the reader may learn by referring to Dr. R. W. R. Raymond's articles in the *Trans. of American Institute of Mining Engineers*, or to those of the author in the *Annual Reports of the Colorado State School of Mines*.

The only remedy is to repeal the present enactment, or

else so prescribe and define the subjects of the U. S. grant, that a purchaser shall have a warranty title to the entry. At present he has possessory right only, and this state of affairs will continue just so long as the present system attempts to convey a right to mineral apart from that to the soil.

It has come to be accepted that litigation is one of the regular and inevitable stages in the development of a mine—the fruit of a strike. Justice W. E. Church, in a concluding and conclusive sentence of a decision said: “The present laws are a hot-bed of litigation and a fruitful source of error.” Judge Bradley declared them “imperfect,” and those who have had any experience will say “Amen.”

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CHAPTER II.

PREPARATORY AND EXPLORATORY WORK.

5. Discussion of the means of reaching veins by shafts, slopes, tunnels, and adits; conditions and comparative advantages; dimensions of the entries. 6. Levels, drifts, and gangways; necessity for, and positions of, reserves; size of lifts and stopes; ratio of dead work to stoping ground; dimensions and extent of gangways; cleats and their influence; mode of finding the continuation of a vein beyond a cross-course or fault; mill-holes. 7. Quarrying and "getting" of salt; hydraulic mining; exploitation of peat and phosphate beds.

5. ASSUMING that the question, "Can the deposit be worked with profit," has been answered in the affirmative, the following features are next considered:

1st. Preparatory works—shafts, tunnels, and drifts.

2d. Exploitation.

3d. Plant organized for hoisting, pumping, ventilation, and treatment.

The means of reaching veins are by shafts or slopes, or by adit or cross-cut, the determinative factors in the choice being local or casual conditions. A blanket vein, without outcrop, is reached only by vertical shaft. But, as most veins crop to daylight, the choice of a mode of access is governed by the engineer's geological knowledge and the system of mining to be selected. Metalliferous veins present the greatest difficulties, because of their uncertainties and irregularities, and at the outset the problem of selecting an entry site is not simple, demanding as it does the best judgment of the engineer.

The entry should be centrally located near the rich ore-body, and in the best position for drainage and underground haulage. Often several entries are operated when the danger of caving may require a hasty removal of mineral. Generally, how-

ever, a mine is planned for a long run, hence its treatment differs from that of a short lease. Much of the success of a mine depends upon the location of the entry-mouth. Concentration of the plant and ample dumping room must be obtained, and boggy ground shunned for foundations.

Wherever practicable, drifting on the vein by adit is favored. The first cost may be, the running expenses certainly are, less than by slope or shaft, and the cost of equipment is *nil*. Occasionally the outcrop of the vein may extend along the hill-slope under such conditions that a series of adits may be driven at convenient distances to explore the vein, and at the same time develop it. But such cases are rare. Each adit then serves for haulage and drainage of its own block of ground. It is then of the customary dimensions and grade.

Cross-cut tunnels have some of the advantages of adits, but more disadvantages. They are run from the steepest part of the hill and the lowest available point, through the country rock, to the vein. They favorably attract capitalists because they serve to prospect and to drain a considerable field and furnish a cheap, secure permanent way. Instances of successful development by this means are few, while the many failures or disappointments are not encouraging; for the vein may be poor and split where it is reached; its grade may be changed with depth or it may be dislocated by a fault, and thus the lode is not disclosed or recognized. And this discovery made after several years of dead work, has discouraged many operators and frustrated the development of many promising veins. After a vein has been opened and its value demonstrated, a cross-cut is justifiable, and may yield large profit on the outlay. Obviously, the size will depend upon the service. Ample double track-way is obtained from a $7' \times 10'$. Many large tunnels serve as haulage canals. Undoubtedly they are a commercial success, but they involve a scheme too elaborate for the individual. Dozens are over two miles in length.

A vertical shaft may be sunk in the country rock to intersect the vein at a certain depth. But the irregularities of lodes and their eccentricities of pitch make this method as uncertain

as the tunnel. If the shaft fails to disclose the vein at the expected depth, considerable prospecting is entailed to find the lode. Even when the vein is pierced, cross-cuts at stated intervals in depth must be drifted from the shaft to the vein. This variety of dead work is very expensive; and should the vein have reversed its pitch, the expense may become a serious item, and the length of the cross-cut required to reach the vein may become unprofitable and deter most operators at the outset. And well it may. It is slower and more laborious than tunnelling, but develops a more economic system and promises surer results unless the mine is very wet. The shaft is safer on the foot-wall than on the hanging side of the lode, but may not always be so advised, for each lower cross-cut is longer than its predecessor, and in hard rock and a vein not steep its cost may soon be prohibitory. Of course these cross-cuts lengthen as the shaft deepens, but the matter of driving is now so quickly done as to cost comparatively little. So this is of minor importance to what it once was. On the other hand, instances of recent wrecks and abandonment from the caving in of hanging-wall shafts are common.

In conjunction with this great outlay is the uncertainty of the continuity of the lode. This may be obviated by sinking on the vein. Inclines are in favor for many reasons. Following its contortions, they explore the vein, and more or less pay their way. Though the cost of maintenance is much higher than shafts, these slopes are preferred in coal regions where the dip is over 10° and the depth not over 500 feet.

When the vein has frequent enlargements, becomes tortuous and even knuckles, or if a fault is encountered, the pursuit becomes awkward. The question then arises as to whether it is advisable to continue the dip, follow the sinuosities of the vein, or begin anew; but the conditions under which it is not advisable to sink a new shaft entirely are very few.

The author favors the plan of sinking on the vein until its value has been demonstrated, after which the slope may be relegated to subsidiary purposes as a second outlet, for escape or ventilation. If, however, the operators prefer to risk the outlay

upon a cross-cut tunnel or shaft at once, they will have the most conservative form of attack if the vein proves good to this level. But considerable development work might have been done from the incline, on the interest upon the investment, while trying to reach the vein. No good exploitation can be effected until the conditions of the vein are developed. For attacking beds which are less freaky this preliminary incline may not be justified, but with the vagarious veins this plan seems indispensable. A subsidiary slope-entry, partially prospects the vein; and so long as two outlets are advantageous it seems rational to first disclose its value before venturing on the tunnel, or the sump shaft and its succession of cross-cuts to the deposit.

There is a diversity of practice as to the dimensions of a tunnel, drift, or adit, varying with the demands upon it. The dimensions of the main level should always be as great as convenient, because of its service. It may be driven in the country rock, as more advisable and safer for a permanent way, and it not unfrequently happens that the country is softer than the lode. For a single stope-lift one compartment suffices for an adit. A tunnel is generally double-tracked, and frequently has an additional compartment for ventilation. The plan of laying two or even three rails in a narrow tunnel, which is only widened at turnouts for four rails, is of doubtful economy. The inconvenience of a crowded gangway is undeniable; the relatively low initial cost is its sole commendatory feature. Yet the difference in cost is not so great as might at first be imagined. In a large tunnel greater advantage can be taken of the face in drilling, and but little more powder is required per lineal foot; the difference in cost of timbering is little, if indeed it is anything, and the cost per cubic yard, broken, is much less than in small headings. Not even an approximate estimate can be given of the progress and cost of driving tunnels. They vary from \$3 to \$15 per cu. yd. of material removed. In granite it cost 90 c. per cu. yd., using 40 per cent Giant and percussion drills; progress, 750 cu. ft. per day. In porphyry 200 cu. ft. can be removed, costing \$1.90 per cu. yd.

The upper bench is driven first, after which the bottom is easily lifted. See Chap. 8, Part II.

Slopes and shafts are of such dimensions as the hoisting, pumping, and ventilating appliances require. Slopes of two compartments are generally 12 to 16 feet wide, increasing to 18 or 20 feet if three are provided for. The height is fixed by the dip and the conveyance employed. Nine feet is not uncommon in a 35° dip, where the car is elevated by carriage. In driving, the lower bench is kept in advance of the rest.

6. The preparatory workings are far from complete when the ore-body has been struck. Permanent gangways for haulage must be run and securely supported. In coal-mines two parallel ways are driven with a rib 20 feet between them, one from each entry. In thick and steep veins the haulage-way is built near the floor, to facilitate loading of the cars. The airway is smaller, and above. For the lower lifts of the mine only one airway need be driven—the intake—if the main level of the exhausted lift, or lifts, be connected and employed as a return-airway. When the vein is reached, or penetrated some distance, it is then divided into blocks, according to the system of exploitation. Gangways pitching slightly towards the outlet are drifted right and left in the ore-body—from 60 to 100 feet apart vertically, in veins, and from 200 to 600 feet “on the rise,” in beds. They divide the deposit into “lifts,” or “stopes.” Adits serve as gangways, as well as entries.

As many of these levels or drifts are run as the necessity for reserves, or the exploitation, may demand or the means of the operators will permit. It is undoubtedly advisable to open numerous and large spaces for attack, thus ensuring steady output without “picking its (the mine’s) eyes out.” They are extended to a natural boundary. Though the relation between the cost of maintenance and haulage, and that of sinking a new entry, may prescribe the limits. For example, a thin, deep bed in good ore, having a strong roof, is worked 2 miles from the downcast. Ordinarily, 3000 feet is far enough. In mines working on ore as uniform as coal, or those in bodies of known extent, only a sufficient number of lifts

need be maintained to control the output. If the ore or adjoining rock is soft or decrepitates, the supports deteriorate rapidly, and induce a continual fear of danger from caves or the evolution of gas, so but few lifts are kept open, and each is worked out as rapidly as may be. The height of the stopes, or the length of the lifts, and the ratio to the thickness of the deposit, depend more upon the ore value than on the method of mining. The distance between the levels is increased with the hardness of the rock, the smallness of the deposit, and the low grade of the ore. The lifts are shorter as the intended output is large and the inclination of the vein great.

How and where to place the level in the lode is of great importance. In the middle or on either side? With a lode of uniformly low-grade mineral it makes no difference. Generally it is safer to keep it in the foot-wall, or along it, if the country is softer. Injury by subsidence is less, and seepage of water is more readily taken care of. In thin veins the foot-wall is cut away to secure height for the car, and in thin beds the roof or floor, whichever is the softer. In thick beds the gangway is in the lower bench. If the mineral is in a small streak, it is followed as it jumps from wall to wall, unless the divergences from a straight line are too great. Otherwise the "level" is continued straight, without regard to minor deviations or rolls, on a grade of 1 in 200. The dimensions depend much upon the nature of the ground and the length of time it is to be maintained.

This class of work is very expensive, compared with ore-extraction, and for this reason is called "dead work." But it is indispensable, as exploratory. Though primarily unproductive, its location bears vital relation to the mine economy. Besides careful timbering, heavy stump and chain pillars of ore are left for support, the mineral of which is only incompletely recovered when the lift is abandoned. Indeed, all permanent ways should be so protected as not to jeopardize lives or the mine. Shafts should be surrounded by from 30 to 60 feet of unworked vein; haulage-ways in beds, by pillars 60 feet wide on either side; stopes, by arches of 10 or 20 feet thick.

A fair ratio of total dead work to stoping-ground opened is 1 to 8. In beds the unworked matter for support nearly equals the amount designed to be mined in the rooms.

All rocks are more or less uniformly creviced. Stratified rocks, for example, have horizontal planes of growth and vertical planes called joints, caused by shrinkage. Some coal-beds, besides the horizontal planes of cleavage, are cut by one set of parallel planes only, others by two sets, producing rhombohedral coal. These cleavage planes are called "cleats."

As crevices facilitate the breaking of rock, so do these "cleats" the mining of coal. In fact, in soft coals of small pitch the direction of the cleat alone may determine the direction of the gangways. In order that the working faces may be against the cleat, the most important drifts are with the cleat. This is not so true in anthracite veins because strong explosives are used. In steep-pitching coal-seams cleat is of less importance than the grade of the haulage-ways. Here the main galleries are with the strike, or slightly diagonal to the rise, the butt headings (see Fig. 10) being nearly perpendicular. They should not be driven far before breaking off the face-entries.

Deviations in the course, or changes of rock, occur in the lode, often so imperceptible as to lead the miners away from the vein. The freaks, horses, "jumping" of the streak, pinches, or faults may have gone unnoticed. A temptation to follow the softer country rock often accounts for "losing the vein." It is of common occurrence. In such event, fresh exposures of the sides of the level should be carefully examined for some distance back, to ascertain the point of departure and its cause. Cross-cuts in the lode may even be necessary.

If a faulting dike or cross-course is encountered, its strike and pitch are noted. After cutting through to the other side the character of the rock is examined. In stratified country the rock encountered should be identified, and its geological position, relative to the ore-bearing stratum, known, thus guiding the engineer. But if the operations are in massive rock, the problem assumes a serious aspect when he attempts to fol-

low the prolongation of the vein beyond the plane of the fracture. It is a matter of record that fully 80 per cent of the intersected veins were heaved, apparently, to the right or left. Those to the right are twice as many as those to the left. Henwood also discovered that the heaving to the side of the greater angle is five times as common as to the smaller angle. In every district may be found a rule for finding the other end of the vein. But it is purely of local application and unreliable. To formulate a general rule out of these numerous and apparently eccentric displacements would seem well-nigh impossible; but Herr Schmidt, in 1810, offered a solution to the problem, which, though not infallible, is the best extant and has done valuable service. "When the cross-course dips away, after going through it, the drift is run along its far wall in a direction opposite to that in which the vein pitches. If it dips toward the mouth, the drift is carried along the far wall, to the right or the left, as the vein dips to right or left." The amount of the displacement, i.e. the distance to be drifted for the continuation of the vein, cannot be premised. It varies between very wide limits, and is thousands of feet in many localities. Henwood averages the throw of veins at 16 feet.

The vein is still further divided into parallelopipeds, by mill-holes 50 to 150 feet apart. Through these the mineral descends to the level, from which they are upraised. Winzes, or secondary shafts, do similar service, but until connection with the lower level is made the mineral is hoisted from the stopes they work.

7. Quarrying is the simplest means of extraction. It disengages large masses, and admits of operations on a large face. It may be employed for all deposits near the surface, when the removal of the alluvial and friable rock is cheaper than timbering them up. Slate, building-stone, iron, lead and zinc ores, peat, coal, graphite, and mica are thus mined. The overlying loose material is stripped, and pays better than mining, so long as not over 4 yards of soil must be removed for each yard of coal. Practically all the deposit is recovered, and to a moderate depth is quite profitable. The point selected for the

beginning of the work, and the discharge of the output, is the lowest convenient spot for transportation. Hoisting is accomplished by derrick and buckets; drainage, by bore-holes and wells sunk deep enough to drain the pit. The influx and accumulation of surface-waters give trouble, which is somewhat relieved by ditches and drains dug alongside the quarry. But the limit is soon reached with the difficulty of propping the unsupported sides of the cave. In the Tilly Foster mine this problem is solved by the blasting away of 200,000 tons of the hanging wall. Such work is dangerous and uneconomical, though it is systematically employed for iron ores all over the world. Nevertheless, the critical moment must come when a more rational method will be necessary. It is difficult, however, to induce a change to the more expensive underground work, and where it must ultimately be adopted the previous quarry is deprecated. Increased pumping-machinery and more timbering will be required than if the mine had not been previously injured. Several properties might be mentioned, in Leadville and Lake Superior, in which vast quantities of ore were lost in the caves primarily caused by originally working as an open pit. Nevertheless, quarrying by the steam shovel is common.

The above strictures placed upon quarrying do not, of course, apply to the extraction of structural materials, which always occur superficially and flat over extensive areas. Building and mill stones are best recovered by open work, and easily mined in blocks by trenches and channellers (see Chap. VIII, Part II).

The getting of salt is generally by a special process. It is always found in old river-beds, and quite liberally distributed over the world. In England and in Germany the thick beds are mined systematically. Elsewhere, the heavy investment of capital involved would militate against the mining of impure beds, especially if cheap fuel is to be had. Then holes are drilled to the bed, water poured down, and the rock-salt leached out. A pump-pipe is carried to the floor, and the strongest brine thus drawn. It will be seen that the capital required is thus reduced to a minimum, and the output may

be increased at a moment's notice. There will be no expense for storage, and no deterioration. This solvent process is also used in mines which have collapsed. The brine is evaporated by solar heat, or boiled in drying-pans. One ton of coal will evaporate 1600 gallons of brine, carrying 77 bushels of salt.

Hydraulic mining is a species of open work, in which water is the agent for removal. The main objection to it is the damage done by the sediment and waste in inhabited regions. An ore of 20 cents in gold per cubic yard pays. A. Bowie's "Hydraulic Mining" is a complete work on the subject.

The exploitation of peat and phosphate beds is by dredging. In heavy bogs of the former, canals are run for drainage, and for the navigation of a scow, which cuts away the peat. Afterwards it is pulped, pressed into blocks, and dried. This furnishes a clean, cheap fuel. Phosphate rock, for fertilizers, is dredged and grappled for, in rivers and deep water, by machines. Peat, or fertilizer above water-level, is quarried in steps.

Materials which occur in large bodies, and regular, require systematic exploitation. Short fissures, feeders, gash-veins, and pockets can hardly be classed as other than special deposits, for which local conditions determine the means of mining. Beds and veins of clay, salt, coal, gypsum, and the metals have a continuity and a consistence sufficiently uniform to admit of classification as minable masses. They are found in all manner of positions, with varying boundaries and variable admixtures of foreign substances. There is always a right and a wrong way of doing things, so it rests with the operator to select the best method of husbanding the resources of his mine.

The cost per ton is by no means the sole consideration. A speedy and complete removal is of utmost importance. Differences in dip and thickness, the relative amount of barren rock in the seam, the amount of gas, and the character of the bounding walls, are the factors determining the choice. Other elements of perplexity are added to the problem, as the friability of the ore, the disintegration of the vein matter, and its

value, but these are of minor import. It should be borne in mind, also, that each method has its special adaptability. Numerous instances of failures may be quoted resulting from the error in the adaptation of a good method to wrong conditions, and it is earnestly hoped that a careful perusal of the following brief conditions may be of assistance to mine operators.

Whatever the method, first, facilitate the breaking of mineral by making the working places large, with ample, free, face; second, concentrate the workmen as much as possible; third, reduce the length and cost of gangways to a minimum, keeping them open, only so long as needed, for stoping and the robbing of the supports.

The steam shovel with its dredger and derrick plays so important a part in the operations of extracting the soft iron ore of Wisconsin and Minnesota as to be worthy of mention among the economic methods of ore-mining. That it has revolutionized the iron-manufacturing industry there is no doubt; for more than 13,000,000 tons of iron have been extracted for the past four years by its aid.

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CHAPTER III.

METHODS OF MINING.

8. Analysis, discussion of the general applicability of mining, "re-treating," differences between coal and metal mining; the least minable thickness of deposits. 9. Overhand and underhand methods, comparison and applicability of; account of the long-wall system; details of the plan; gob roads and their care. 10. Pillar and stall method of mining; dimensions of rooms and of pillars; creep, cave, crush, or squeeze, and their prevention; order and manner of winning pillars; mining loss and waste. 11. Modifications of the pillar and stall system; the "County of Durham;" the "Wasmuth;" barrier pillars; relative merits of long-wall and pillar and stall; panel system; "square work;" gallery and pillar." 12. The American system of "square sett," as applied to veins and beds; modes of mining thick seams, in slices or by filling or caving; traverses with filling or with caving.

8. DEPOSITS containing organic, earthy, or metallic minerals may be flat or steep, thick or thin, and accordingly the systems for their extractions are :

Dip less than 45°.	{	Under 6 ft. thick.	{	LONG-WALL	friable or soft roof.
				PILLAR AND STALL.	firm ore.
		Over 6 ft. thick.	{	FLAT STOPES.	
				PANEL	gaseous coals.
Dip exceeding 45°.	{	Under 8 ft. thick.	{	GALLERY AND PILLAR.	hard ore.
				METHOD OF CAVING.	
		Over 8 ft. thick.	{	" " FILLING.	yielding vein-matter.
				SQUARE SETTS.	
				" " WORK	medium-firm ore.
				OVERHAND STOPING . . .	firm vein-matter.
	{	Under 8 ft. thick.	{	UNDERHAND " "	
				TRAVERSE WITH FILLING	friable ore.
		Over 8 ft. thick.	{	" " WITH CAVING	
				SQUARE SETT.	soft ore.

The most simple and natural method would appear to be one in which series of working breasts, as wide as the nature of the roof or walls would admit, are opened after the main headings have been carried to the extreme boundaries, leaving the mineral intact except for the necessary roads. The mineral would then be mined homeward toward the shaft as rapidly as possible. This plan of *retreating* presents the maximum of safety, for all the miners are working in solid mineral and of economy, for the operator may win all of the deposit. This plan requires great patience on the part of the operator, and certainly calls for large capitalization, so that if the area or the royalty be large it cannot unconditionally be advised. But the cost of mining is far less and the product per acre greater, whatever the system, than by mining toward the boundary.

The mining of coal-seams, earthy beds, and of metalliferous deposits are not dissimilar, except that tentative explorations are necessary to ascertain the nature and extent of the latter, while the greater regularity of the former enables systematic development to be proceeded with at once. Coal-seams are less complex and more easily mined than are other beds. Special attention, however, is given in collieries to ventilation, and to the demands of the coal-market for the large sizes, which require great skill and care for their production because of the brittleness of the mineral and the weakness of the floor. The value of metalliferous ores is not depreciated by the fineness of the mineral.

As to the minimum thickness of minable veins, local conditions govern. The smallest coal-seam known to the author to be "paying" is 28 inches thick; the standard is placed at 1 ft.; and 43 inches is the thinnest iron ore-vein.

After the steep lode has been blocked out, such blocks as are deemed workable are mined by one of the two methods—overhand or underhand. In the first mode of working, the miner picks or shoots down the ore in front of and above him, advancing as high as he can reach a breast, which, when exhausted, is followed by another similarly mined above him

Notwithstanding the imminent danger from falling rock, it is a less arduous, quicker, and cheaper method, because gravity facilitates all operations of breaking away the vein-matter. In the underhand work he removes the mineral below him and progresses the breasts longitudinally with the lode, taking them successively downward. These methods are applicable

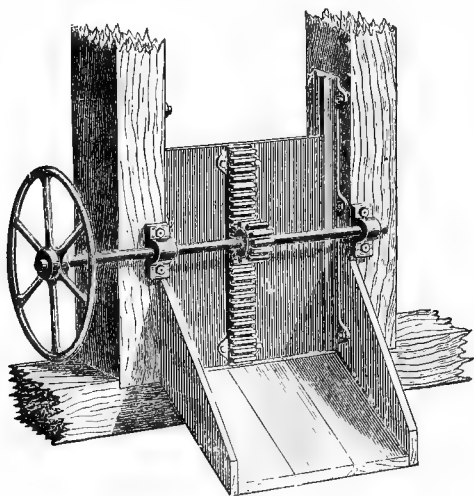


FIG. 1.

to steep veins, though the overhand working is equally advantageous in thick flat beds.

9. UNDERHAND stoping is possible under limited conditions only. The several blocks of the vein which are to be attacked are worked away in horizontal slices, beginning at a winze (Fig. 2) and proceeding downwards. The miner standing on the floor of the level removes the vein-matter for a depth of 6 ft., picks out the ore, and throws on a platform behind him the waste. This slice is carried as far as he designs going—usually for one half the distance between the winzes. As each slice is removed, its ore is raised to the upper level, while its gangue partially covers the platform replacing the vein. If the entire vein-matter is “pay-dirt,” the staging may be dispensed with.

In OVERHAND stoping the attack begins at some point on the lower side of the blocks (Figs. 3 and 4). The miners stand on the caps of the drift-timbers and remove a horizontal slice as wide as the vein, or the streak being stoped, and 5 or 6 feet high. The length of the slice is rarely over 60 feet. The ore is picked and delivered to the chute, which is carried up with the progress of the stoping, while the waste rock is piled on the gangway timbers behind the men. The next upper slice is broken away in the same manner and direction, beginning with the mill-hole, the miners standing on the "gangue" of

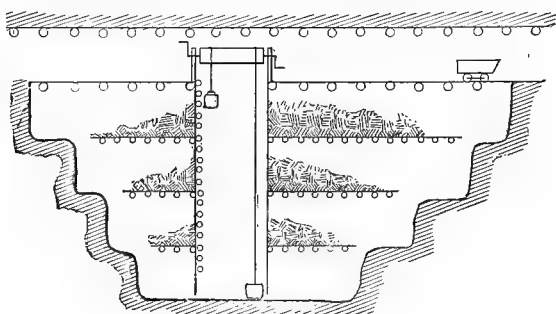


FIG. 2.

the previous slice. If, however, there is not sufficient gangue, a temporary staging is placed at convenient height; or the rock from the walls is broken away to supply the necessary filling. Again, if the total vein-matter is salable, enough of the broken ore is left upon the stulls for the miner to stand on while drilling, and only the excess is hoisted at once until after the full stope has been raised to the upper level, or to within 10 feet of its floor.

Another plan consists in having one gang of miners to each slice and in mining the several slices of a stope simultaneously, the breast of one being 10 feet in advance of that next above. This mode of working presents the appearance of inverted steps, and hence is called *Overhand stoping*. Fig. 5 represents a mine worked by overhand stoping, the white spaces being unworked ore, the total cost of mining, transporting, and treating the copper ore being \$1.19 per ton.

This method is occasionally called "flat stopes" when used in coal-mines having a pitch of about 50° . Except for the name, it is identical with that described.

Taking the amount of timber used as a criterion, the *Overhand* has the preference; for it requires no timbering other than the stull for the reception of the waste and the protection

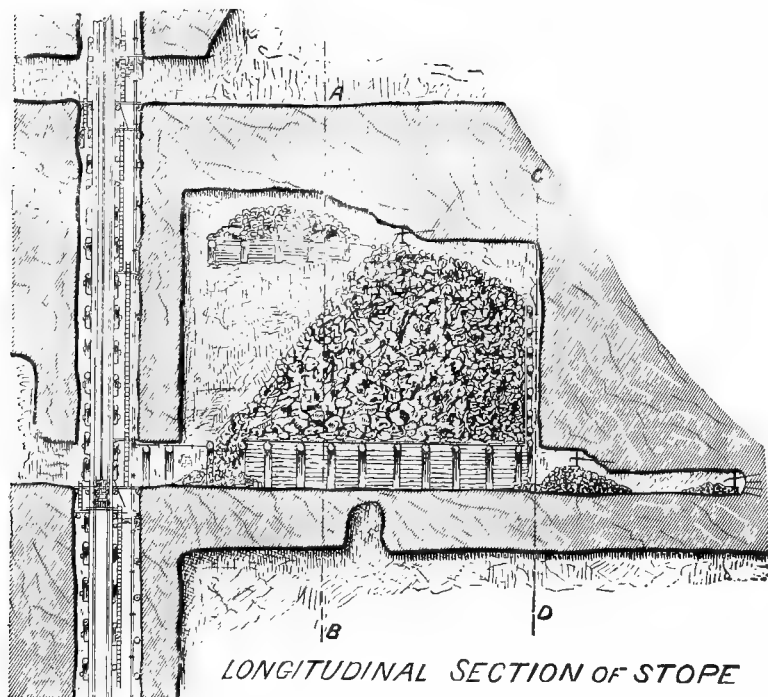


FIG. 3.

of the gangway, while *Underhand* requires as many platforms as there are stopes. The former affords greater facilities for breaking down rock, and is employed to advantage in a wide lode. The width of a lode capable of being worked by *Underhand* is limited to the length and size of the stulls necessary and available. It is accounted best for valuable ores, because all the rock must be handled, and none need be lost; but be-

cause the ore is continually being trodden on, it cannot be used for brittle ores. In the *Overhand* the vein-matter is shot down on the stull dirt, and unless the latter be covered by canvas, bull-hide, or boards, the loss of fine material is inevitable.

The LONG-WALL system is very simple, and in general use for coal, iron, clay, etc., in thin seams, and also for thick seams having partings. It contemplates a complete removal of the deposit without making any attempt to support the roof.

Beyond the solid portion left intact for the security of the hoistways, the attack upon the deposit begins after the two

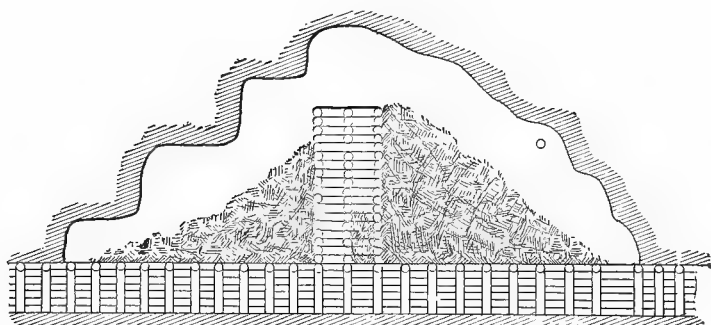


FIG. 4.

or more parallel ways are driven (page 24). At convenient distances apart, tram-roads are driven to the rise of the seam, the entire wall of the vein being simultaneously attacked. From each road a gang of men, called holers, undermine the face of ore (see pages 409 and 480), and at sides of the holing make a vertical cut or shear. This is left over night to break down by the pressure of the overlying strata, or it is propped up until another set of men, called getters or loaders, take it down by wedges, picks, or explosives. By this means "large coal" is obtained, the slate or refuse being thrown behind the men into the excavated space or "gob." Meanwhile the packers and timbermen follow, keeping the working places secure, and the roads which are in the gob well-timbered. At dis-

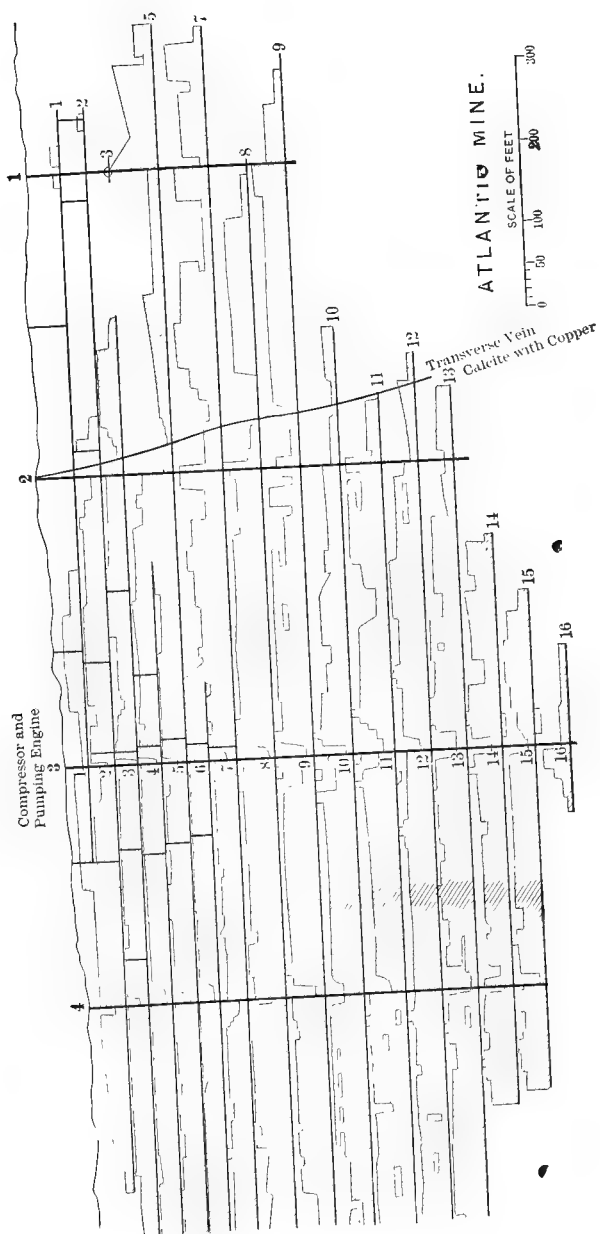


FIG. 5.

tances depending upon the strength of the roof, props and head-pieces (Fig. 8) are set in two lines for 10 feet back from the face. With a good roof one line may suffice, the props being advanced as the face progresses. Under a poor roof they are set closer together.

The face of attack may be of the horseshoe form (Fig. 7) from the gangway; it may advance elliptically from the bottom of the shaft, if the pitch is very slight; or the working-faces may present the appearance of flat stopes if the dip is as much as 30° . The last-named plan gives more cutting and greater friction to the ventilating current, besides offering an opportunity for the roof to crush off from the corners much coal that is lost.

The ore may be conveyed from the face over a tram-road

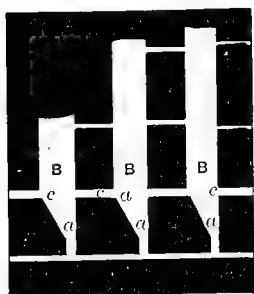


FIG. 6.

laid along it and into the "gob-roads," or it may be directly delivered to cars travelling only along the gob-roads. In the latter case the roadways are smaller and more numerous than is necessary in the first-mentioned case, which is only possible with a tough flexible roof. When the roof is brittle and weak there will not be space enough for a car along it because of the props placed close to the breasts.

The width of the faces depends upon the nature of the roof, but it varies from 120 to 180 feet with a roadway leading up to the middle and a gang of holders on each side. While the yielding and the fracturing of the roof is the one condition essential to the success of the method, it also constitutes a serious objection. The method contemplates a subsidence of the roof behind the miners and on either side of the roads, to supply material for filling the gob. This tends to close up the roadways, and for their preservation not only are they protected by building pack-walls (Fig. 9) of from 4 to 10 feet thick out of the roof-stone, but the roof must be constantly hacked or the floor cut to give head-room. As their number and length in-

PLAN OF LONG WALL.

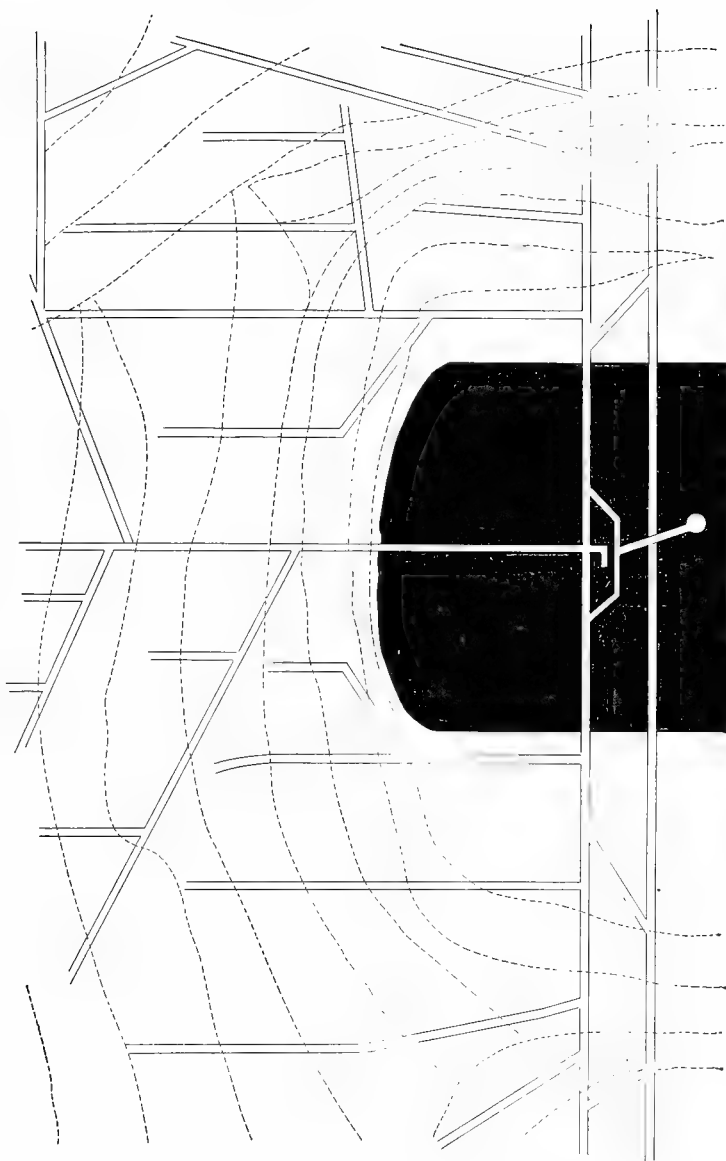
Dotted Lines, Progress by Months.

FIG. 7.

crease with the progress of the work, their cost soon becomes so serious an item of expense as often to open the question of the desirability of continuing the plan when the boundary is a great distance away. The experiment of leaving a thin rib of unworked mineral on each side of the road failed. The safety of the mine and of the men depends upon the security of the

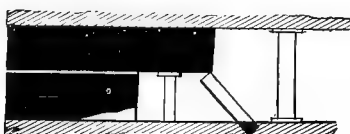


FIG. 8.

gob-roads; and if the pack-walls cannot cheaply be made efficient, or if the operators will not exercise the courage of their convictions and mine *retreating*, operations must then cease until the mine is divided into several long-wall sections with separate hoist-ways, or else some other system is employed.

In collieries there is an additional source of danger in the gob; i.e., the liability of the stowed waste to spontaneous com-

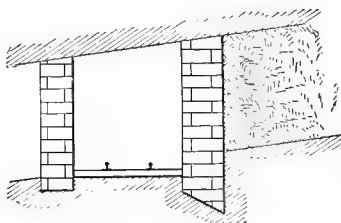


FIG. 9.

bustion. Though plastering the walls may delay the fire, the slightest chink admitting air undoes the work, so that the only surety against it is to fill the gob with clean rock only, if at all.

10. The system known as the PILLAR AND STALL consists in driving long, narrow working breasts ("stalls" or "rooms") between solid pillars. It is the most common and, perhaps, the oldest system for mining flat beds of any variety of mineral.

After the gangways have been carried a safe distance beyond the entries, the rooms *B*, Fig. 10, are turned as fast as possible to the rise, or backward up the slope, Fig. 15, according to the mode of conveyance of the ore (see p. 56), from breast to gangways. The jaws, *a*, of the rooms are 6 or 8 feet wide for a distance equal to that allowed for the stump-pillars, *A*, when they are suddenly enlarged to a working face of 20 or 30 feet, as time and the condition of the roof will allow. The rooms are carried, regular and uniform, to a length of from 8 to 10 times their width, leaving a chain-pillar for support to the upper level. The rooms of each lift are mined progressively to the boundary as rapidly as circumstances permit.

The cost of driving narrow entries and headings, marked *a*, *a* and *c*, *c* in Fig. 10, is so high, that operators have introduced a modification whereby the jaws *a*, *a* are driven only 10 feet long, after which they are gradually widened out to the full face of the room (Fig. 6). By this means the amount of "narrow-work" is reduced and the cost of opening rooms materially lessened, while the pillars are not weakened.

The pillars, *P*, left between the rooms are unbroken except for three or four small connecting-passages, *d*, *d*, through which the return ventilating air-current passes from room to room on its way to the upcast.

The relative dimensions to be given to the pillars are determined by an answer to the question: Are the pillars intended only for the temporary support of the roof, or are they to be regarded as reserves to be mined at an appropriate time? The pillars conceded for the former purpose are thinner than those to be maintained for future supply. In any event, the pillars constitute the support to the roof while mining is in progress in the adjoining stalls. Their dimensions must be such that each will not only support the mass directly above it, but also resist the crushing transmitted to it by the roof over the room. Every square foot of roof area receives a pressure of 8 tons per 100 feet of overlying strata. Thus, at 700 feet depth the rooms are 20 feet wide, and each square foot of pillar will be subjected to a weight of about 90 tons per square foot—a big

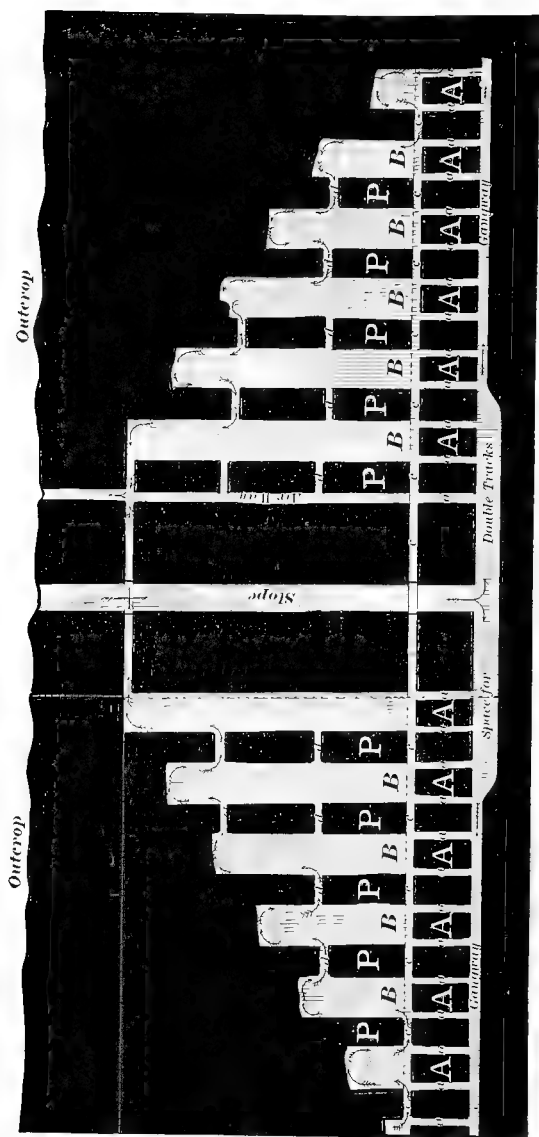


FIG. 10.

load even for stone pillars. As the crushing resistance of the different ores and coals is not known, no formula can be offered for determining the size of pillars, and each mine will have to be guided by the tests in its own locality.

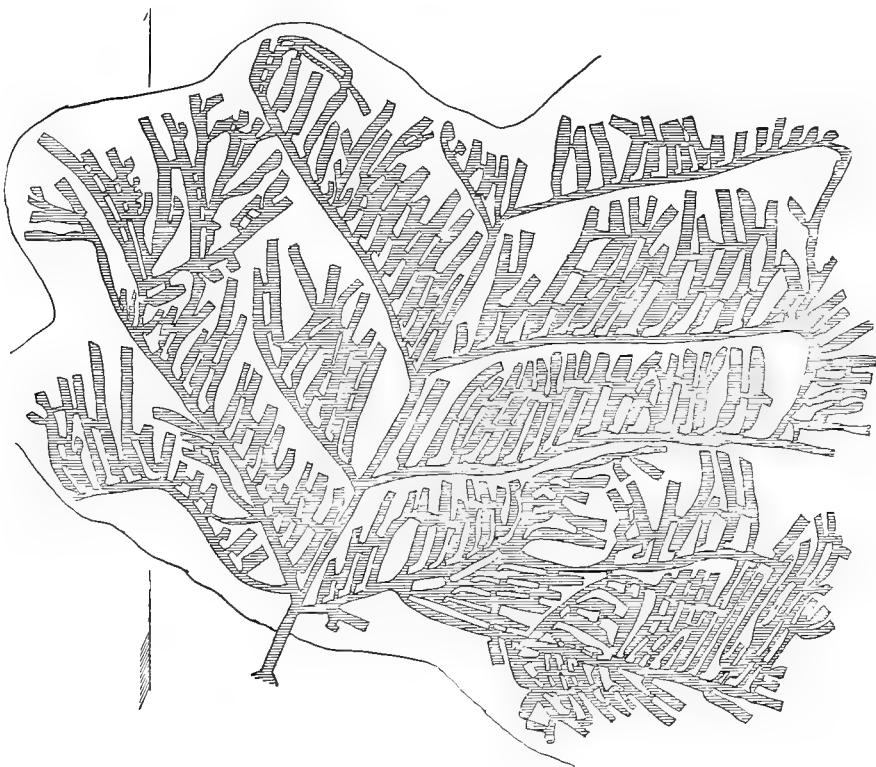


FIG. 11.

Though the resilience of the strata above may relieve the roof of much of this weight, the size of the pillars will increase and of the rooms decrease as mining deepens, until a limit of workable depth is reached, when the rooms are reduced to an unprofitable portion of the deposit. At a depth of 700 feet they contain less than 30 per cent of the total mineral; at 1600

feet the pillars leave only 20 per cent of ore to the rooms; while at 2000 feet the rooms will be only one third as wide as stall-pillars, and capable of producing not over 15 per cent of the total contents of the seam.

Large pillars are indispensable under whatever condition of roof or mineral. A good roof admits of wide rooms, but it also requires a thick pillar to receive the thrust of the roof; less pressure will be transmitted to the pillars if it is weak. Stronger pillars are therefore required with a firm roof than under a weak roof, in thick beds than in thin seams, in free or brittle coal than in compact or tenacious coal, and in seams that are solid than where partings exist. Pillars should have a large base when the floor is soft, as otherwise a "creep" would be induced, choking up the excavations, and particularly the air-ways. The rooms are comparatively narrow when the roof is firm and flexible, or a "squeeze" would follow its yielding. But a creep is more serious and insidious in its destruction. Once begun, it is amazing how rapidly the trouble spreads to neighboring rooms. The first sign by which it may be recognized is the wavy condition of the tram-rails; a dull, hollow sound heard when treading the pavement is also a sure alarm.

In addition to the excessive pressure which injuriously affects the quality of the coal, the exposure to alternations of humidity and temperature reduces the resistance and shortens the life of the pillars. Hence, for reasons of safety, the rooms should be mined, the pillars robbed, and the lift abandoned as quickly as possible—mineral other than coal or clay admitting of the expense of a partial replacement of the pillars by masonry, filling, or timbering; and this is advised, particularly in thick beds, where pillars are less efficient than in thin seams.

After all the rooms of a lift have been mined to the boundary the pillars are robbed, *retreating*. This is the most dangerous part of mining. Scales are removed from the sides, the end of the pillar is sliced off, a heading is driven through the centre as a room, and some of the middle portion of the mineral recovered, or the pillar is worked *Longwall*, parallel

to the strike. By maintaining a regular line of retreat and keeping the faces small, a mastery of the subsidence is retained.

II. Local modifications of this general method are numerous, particularly in collieries. For example, in soft coal the breasts, 20 ft. wide, are first separated by pillars 52 ft. wide between them, and when they have advanced 50 ft. up the slope, the pillar is pierced for another room 20 ft. wide, leaving 16 ft. of pillars on each side.

Again, the empty rooms are filled with waste, which supports the roof while the pillars are recovered.

In the *County of Durham* variation—a connecting link, as it were, between the *Pillar and Stall* and the *Panel* method—the breasts, with their pillars, are laid in groups of 8 or 10 from the gangway. Each such section is then mined separately and systematically toward the boundary. Between the sections large blocks of coal are left, 150 ft. thick, as barriers, unbroken even for air-ways. These give complete isolation to the section, and localize any movement of the roof. If possible the barriers are placed in poor-quality coal, and of a liberal size to relieve the breast pillars. This plan is commendable in gaseous mines as safer than *Pillar and Room*, and more economical than *Panel*. The barriers are not robbed until after the adjacent sections have been exhausted.

The *Wasmuth* is a modification for hard, thick coal.

Some very thick, steep veins of copper and of coal are worked by *Pillar and Stall*. The vein is divided into lifts of about 30 feet high, by laying drifts along the foot-wall of the lode, from which are driven stalls across the entire vein, 20 ft. wide and 18 or 20 ft. high, leaving pillars of the same width between them. The rooms are taken in three or four ascending slices, each of 6 or 7 ft. high. The amount of ore left standing in the pillars, and the small height of the lifts with the consequent dead-work, militate against the application of this method in hard ore. By subsequently filling the rooms with waste the pillars may be recovered.

Whatever the modification of this method may be, it is

certain that the amount recovered from the pillars is small and the culm loss is great. Dr. H. M. Chance estimates a loss of 55 per cent in mining thick, deep, soft coal, and at least 35 per cent in thin, shallow, hard coal, under a good roof. As larger pillars will not give a very much greater product per acre, it is not surprising that the *Longwall* is gaining in popularity.

The relative advantages and disadvantages of *Longwall* and *Pillar and Stall* are easily stated. The expense of maintaining roadways in the former is higher; the number of accidents, greater; the product of round coal, larger (as 68 per cent is to 48 per cent of the area mined); the product per acre, higher; the ventilation, simpler; the amount of narrow work, less; and the consumption of powder in hard ore, less than in the latter system. On the other hand, the latter is more advantageous when faults and dykes are encountered; if the upper strata are wet; if the boundary is at great distance; if the mines are at small depths, or if the surface land is valuable.

Gaseous mines are exploited by the *PANEL* system. Its development is slower than that of the *Pillar and Stall*, but this is rendered necessary by the fiery dangers. The design is to subdivide the mine into panels by driving at suitable distances apart from the gangway, roadways to the upper level, with horizontal roads and breasts to the right and left from them (Fig. 12).

The panels are separated from one another by heavy barrier pillars isolating them completely. After the breasts are mined and the roof has settled, the barrier- and stump-pillars are drawn as rapidly as possible.

SQUARE WORK is another modification of *Pillar and Stall*, by which thick beds of salt, gypsum, coal, puzzolana, etc., are mined. Rooms are opened from the gangway about 150 ft. square, with pillars 30 ft. thick between them. Instead of mining the rooms over their entire face at once, they are divided by two sets of cross-galleries, 20 ft. wide, which leave 9 pillars

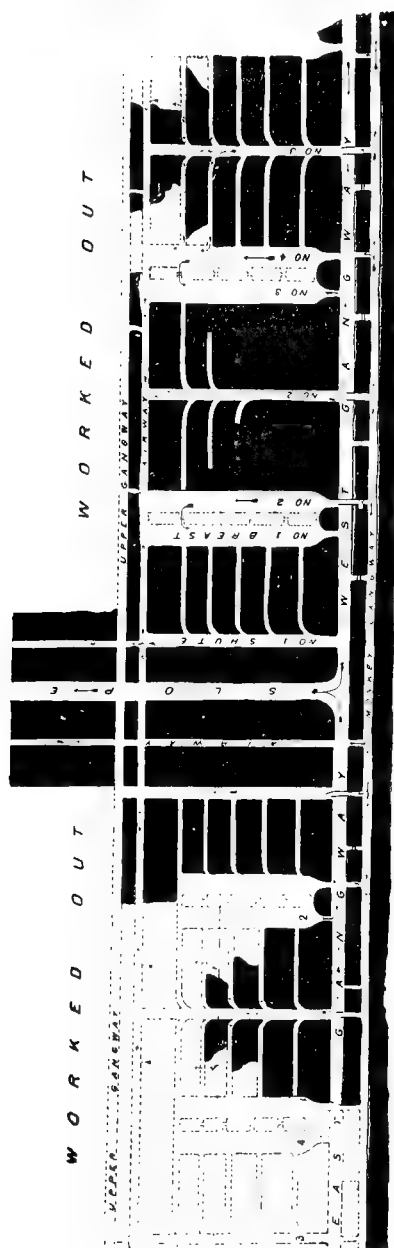


FIG. 12.

about 25 ft. square for support of the roof. The galleries are driven as high as the vein-matter will allow, and if the roof or floor is not firm, a layer of mineral is left for greater security. Though much in vogue, the method involves great risk of life, since the ear, not the eye, must be relied upon to detect a threatening roof. Systematic ventilation is difficult, and, as may be expected, little of the pillars is recovered.

In the Petit Anse salt-mines the galleries are 40 ft. wide and 25 ft. high, while the pillars are 40 ft. diameter.

When filling is inadequate or expensive, the method of GALLERY AND PILLAR may be employed for thick deposits of low-grade mineral. The deposit is mined by driving galleries, as numerous as admissible, and as large as the nature of the roof will allow, and extracting such ore as may be obtained. It is however very wasteful, as the ore in the pillars between the galleries and the sills overhead is entirely lost. Large irregular masses of mineral are mined in this manner.

12. A distinctly American method, known as the SQUARE SETT system, possesses the peculiarity of an equal adaptability to both flat and steep deposits, and succeeds as well with soft as with firm ore. The rooms and pillars alternate, and are of the same width, 20 to 30 ft. The breast of each room is divided into working faces of about 7 ft. wide; as fast as the miner has advanced the breast 7 ft., a square set of timbers (Fig. 175) is built to uphold the roof, and is framed to those on either side of and behind him (Fig. 13). When this horizontal slice has advanced to the hanging-wall or to the boundary of the ore-shoot, another slice is mined in similar manner for a height of 7 ft., with its square setts directly upon those previously placed. The mining proceeds by slices ascending to the roof, after which the pillars may be robbed. The timbering is permanent, and little, if any, is recovered, even if filling is resorted to for additional stability.

In addition to the fact that the expense of timbering is very high, the main argument against the method is the difficulty of obtaining a perfectly rigid framing. The slightest yielding of

props, decrepitation of ore, or movement of walls will result in the burying of ore and its irretrievable loss. (See Part II, 68, Figs. 175 and 196.

Though the methods thus far explained are for the exploitation of deposits of medium thickness, it is also possible to

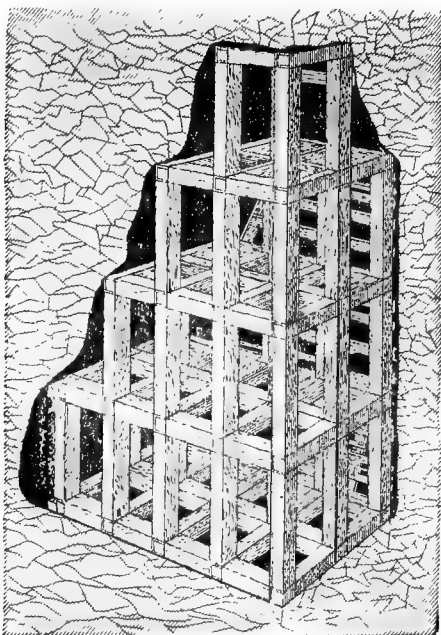


FIG. 13.

employ them on thick veins with equal economy, by taking advantage of the numerous partings, cleavages, or layers of clay that subdivide the mass. Each flat streak is then mined by *Longwall* or by a *Filling* method, and those of high pitch may be worked by *Overhand* stoping. The lower benches, or streaks, are invariably taken in ascending order and advancing, but the top one is mined retreating. Unless stowing or filling is used, the subsidence of the upper benches will result in loss of mineral.

If it is not practicable to subdivide the deposit, it may be mined by some modification of the methods known as *Filling* or *Caving*. The permanent ways—tunnels, cross-cuts, shafts, etc.—are put in the country rock, and levels are driven along one or both of the walls, from which timbered traverses cross the entire vein, having an upraise at the foot-wall end, and a mill-hole along the hanging-wall, or vertically, to the upper level. The deposit is then blocked for attack.

It is evident that, in order to remove the entire deposit, either the roof must be allowed to cave and fill the excavation, or its subsidence must be prevented by extraneous means—timber square sett or rock-waste filling. The plan first mentioned, of allowing the overlying material to cave behind the miners, known as the method with *CAVING*, is very convenient and quite economic, provided the surface land be of little value and the mine deep and dry.

When the preparatory works are completed for two lifts, the block nearest to the boundary is mined underhand from one traverse to the other and across the full width of the vein. When the roof has settled and packed, new longitudinal drifts and traverses are carried in the slice below under the packed roof, and the ore is removed. While this second slice is progressing the first upper one of the next block, in the direction of the shaft, is in process. After these have been worked, No. 3 of the farthest block, No. 2 of its neighbor, and No. 1 of the next, are simultaneously worked, so that the blocks of the level are exhausted in receding order *from the boundary*.

The levels are 60 ft. high, the traverses are about 30 ft. apart, and the slices from 8 to 15 ft. high, according to the friability of the vein-matter.

In very wide veins of brittle rock the traverses are driven 6 to 10 ft. wide, alternating with pillars of equal width. The timbering is cautiously removed from the traverses beginning at the hanging-wall. The vein-matter creeps down into the excavation, and the ore is taken away as quickly as possible. The pillars are sometimes robbed after the subsidence. When

quiet again reigns another set of traverses are attacked in the next lower slice. The stopes are exhausted in descending and retreating order. Planks are sometimes laid on the floor of the workings to separate ore from the débris of the roof, and likewise to afford protection to those engaging in mining below. The consumption of timber averages a cubic foot per ton of ore extracted, not much of it being recovered. See Figs. 14 and 15.

Extreme caution is necessary to prevent sudden falls of roof, and the adoption of the method is only justified where there is an absence of filling material and a scarcity of timber. When the vein-matter is comparatively firm, the method with filling presents greater security, and may be cheaper. In such event the choice of method depends upon the relative expense of quarried rock and framed timber. Timber is neither durable nor abundant. Its framing and placing must be faultless. Clean rock can usually be had in quantities, and may be easily and immediately stowed into the excavated spaces, so the method of *Filling* is supplanting the method of *Square Setts*, because of the perfect security given to the mine.

Undoubtedly, the method of greatest general utility is that of *FILLING*. The plan of the preparatory works (Fig. 236) is identical with that of *Caving*. The timbered traverses *B, B* are driven from the gallery *A*, to the hanging-wall, about 6 ft. square, and maintained until all of the ore has been extracted. These are from 60 to 80 feet apart. If the ore is soft or the vein very tortuous, the gallery is driven straight; and in the country-rock, to render it secure for haulage. Winzes and mill-holes connect the two levels.

There are three modes of removing the ore, depending upon the extent of face which is self-supporting. These may be designated *Narrow traverses, A*; *Broad traverses, B*; and *Longitudinal traverses, C*.

A. From the gallery traverses, *D, D*, 6×6 ft. in area, are driven on either side and adjoining *B*, while its ore is being removed; the excavated space is filled with gangue or rock-waste,

lowered down the shaft, hauled to the winze of the level above, and dropped to the bottom, whence it is spread and compactly stowed in the narrow room to its roof. The next two traverses *E, E*, are exhausted and filled in like manner, until one horizontal slice of vein 6 ft. high has been mined.

A new gallery is driven above *A*, from which other traverses are successively driven and filled,—with this difference, that *b* is not timbered as is *B*, but is opened and closed like the others. The miners stand on the waste while breaking down the undercut ore, and fillers spread behind them the material which is being supplied through winzes from above, while the ore is dropped through the mill-holes to the lower level. Winzes are gradually closed and the mill-holes extended with the upward progress of the mining. Unlike in *Caving*, the slices are removed in ascending order.

This plan is used for thick, highly inclined seams of coal as well as of copper. In some lodes the traverses *D, F*, etc., are not taken until after the alternate traverses *B, E, G* have been filled.

B. When the vein-matter is dry and stands well, it admits of a working face wider than 6 ft., and the attack is made from the foot-wall gallery for a width of one half, or even the whole, of the block between the permanent traverses *B, B*. The ore is delivered to them, and thence out, while filling from above is spread behind the men, who mine across the vein.

C. This is for veins of not over 20 feet width. Here the face of attack is the sides of the pillars. The miners on either side of the pillars from *B, B*, advance toward each other longitudinally with the vein.

This method of mining is very cheap. The rock may be delivered below for 30 cents per ton, very little timber is required, and the breaking down of the ore is very simple. Except that portion of the vein in the lowest slice, which is taken "off the solid," all the ore is undercut and free on its lower face, and hence is cheaply broken. It has been found that the cost in *B, D, b* and *d*, respectively, is as 1 : 0.60 : 0.40 : 0.30.

Any material will serve for filling, but the best is that mixed with clay, as it packs well. It should be in as large pieces as may be conveniently handled, for the "smalls" settle too much. A few weeks' time is sufficient to compact the entire mass of filling, so that no difficulty is experienced when the stowing of the upper level is reached.

This method is indispensable where the entirety of the vein is to be removed, and as it is perfectly safe and much cheaper than *Pillar and Stall* or *Caving*, it gains in favor wherever tried.

The mode of delivering the product from breast to entry and of transporting the same is a most important problem in mining. It usually determines the plan of operations.

If the veins are vertical, or nearly so, the problem is very simple. The ore is dropped through mill-holes to the cars in the level below (see Figs. 3 and 4), or it is hoisted through winzes (Fig. 2). Until the pitch becomes less than the natural slope of broken rock (40°), the ore will roll or slide on the rock bottom of a chute. The sides may be built of plank spiked to props reaching from floor to roof; a platform is built at the lower end of the chute, whence the ore is emptied into the cars or the mill hole is closed by a gate (Fig. 1), which is manipulated by the trammer. By keeping the chute full of ore and only emptying the excess through the gate, the loss from attrition is very much diminished. At a grade much flatter than 40° , the chute must be floored with wood or with iron; but soft ore will neither slide nor roll at less than 18° . If the working-room is large, the breast wide, or the vein thick, the chutes are carried up with the work, one on each side, with waste filling or timbering between. In small rooms, the chutes are built centrally with the manway compartment alongside of them. In some mines with large rooms or stopes, a gravity plane, similar to that shown in Fig. 69, is used as an expedient more or less temporary.

All those beds which lie at an angle between 18° and 6° from the horizontal are the most difficult to operate. They are too flat for self-acting transportation and too steep for any cheap system of haulage. An endless rope or plane for each

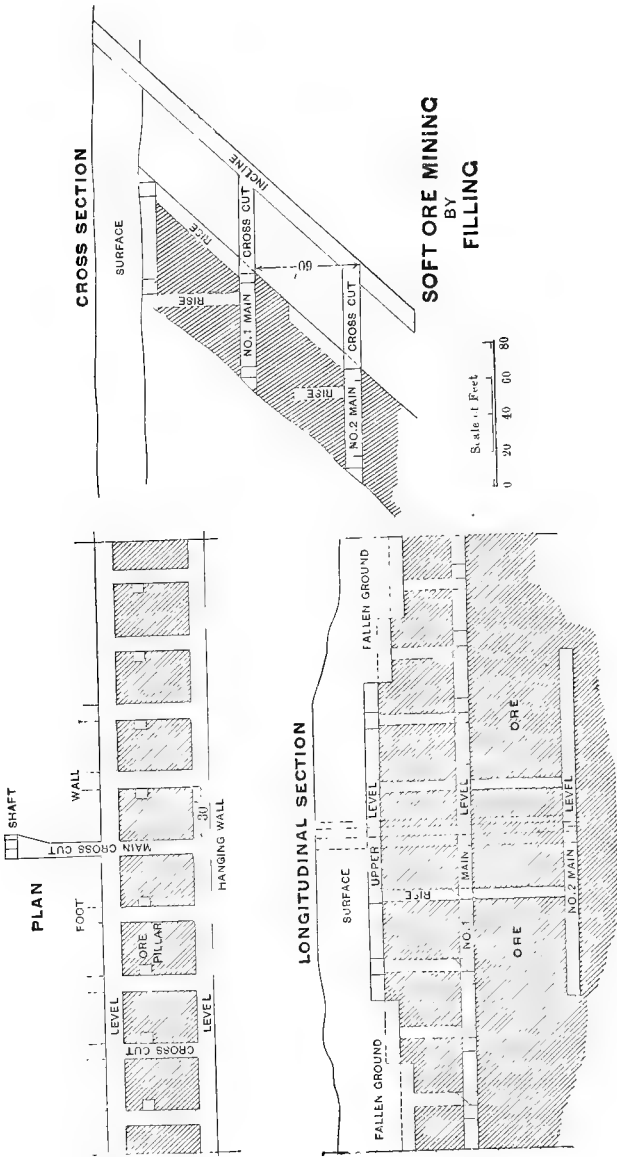


FIG. 14.

room is too complex; and as any scheme involving shovelling would be both expensive and injurious, especially to friable

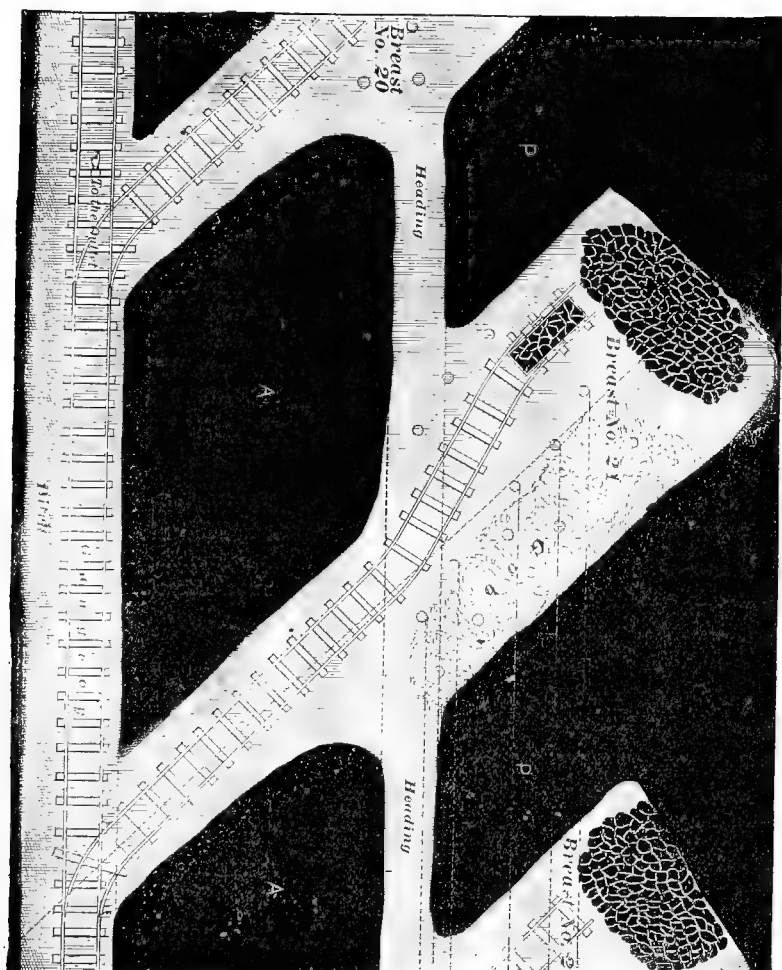


FIG. 15.

ores, buggy roads are resorted to for the rooms. Two lines of props, 6 or 8 feet apart, are held together by thick planks across the road, on which are laid longitudinal stringers to

support cross-ties for the rails. The grade of the rails may be the same as that of the room, or it may be moderated by raising the lower end of the track.

Breasts on a grade of less than 10° are said to be flat, and the accessory haulage from them presents no special difficulty. The rooms are not driven directly up the rise, but diagonally (Fig. 16), and toward the outlet, to secure a favorable grade. Here mules are utilized for power, as the rooms are usually quite long (see p. 24). Tramways follow the advance of the breasts, and are then called "runs." With a bad roof, the track is laid near to the pillars.

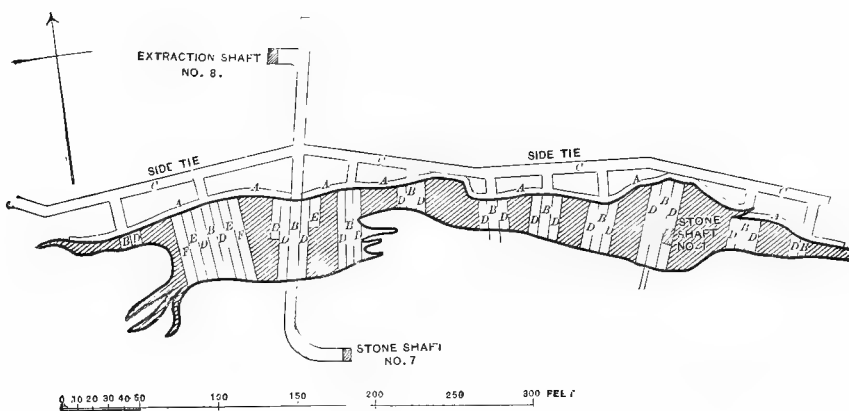


FIG. 16.

Rooms in beds as flat as 3° or 4° are driven "up the pitch," utilizing mules or men as motors to the cars which are taken to the breasts.

It is rare that rooms are worked downward on the dip, unless the pitch is very slight, or the seam is undulating, as steam-power would then be necessary for haulage.

When a flat bed is free from "troubles" and faults and dips slightly, there should be no difficulty in having a complete and regularly outlined system of workings and of haulage, though it is also true that the seam of medium dip gives the greatest haulage difficulties.

The method of mining deposits that occur in large masses

depends upon their extent and form. Large regular ore-pockets are worked in vault-like chambers of a size as large as the firmness of the ore admits, with pillars directly over one another. If the vault should be very large, a modified method by square work (p. 46) will serve.

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CHAPTER IV.

HOISTING MACHINERY.

13. Manual labor; description of windlass and winches; the work of man; examples; modes of increasing the efficiency of a windlass, double and conical barrels. 14. Hoisting by horse and whim; the work of a horse; examples; descriptions of whims, derricks, pulleys, etc.; double and conical drums. 15. Engine hoisting; conditions, etc., for selecting a machine plant; sectional and tubular boilers and their care; consumption of fuel and water; anti-incrustators and economizers; importance of the concentration of machinery; distribution of power; location of hoisters; description of the engine; cut-offs and condensers. 16. Descriptions of types of hoisting-engines; first- and second-motion engines; gearing and friction hoisters. 17. Description of the various types of friction-clutches; drums, their sizes and construction; the Calumet and Hecla leviathan; modes of equalizing the work of the engine; conical drums, reels, and counterpoises.

13. ADIT or tunnel workings are growing more rare, and it becomes an absolute necessity to deliver to shafts all products of the mine (ore, water, or waste), and this traffic is an important matter.

For shafts of moderate depth, and for winzes underground, manual labor on a windlass is employed, though the amount that can be raised by two men 100 feet in eight hours cannot possibly exceed 4 tons, allowing for delays, etc.

The windlass has a cylinder 6 to 10 inches diameter, long enough to reach across the shaft, resting by its axle on up-rights, and operated at each end by cranks 15 inches long, and set at right angles to each other. Iron crabs or winches are to be had in every possible combination. But the simpler the machine the less is the friction, and the more acceptable it is.

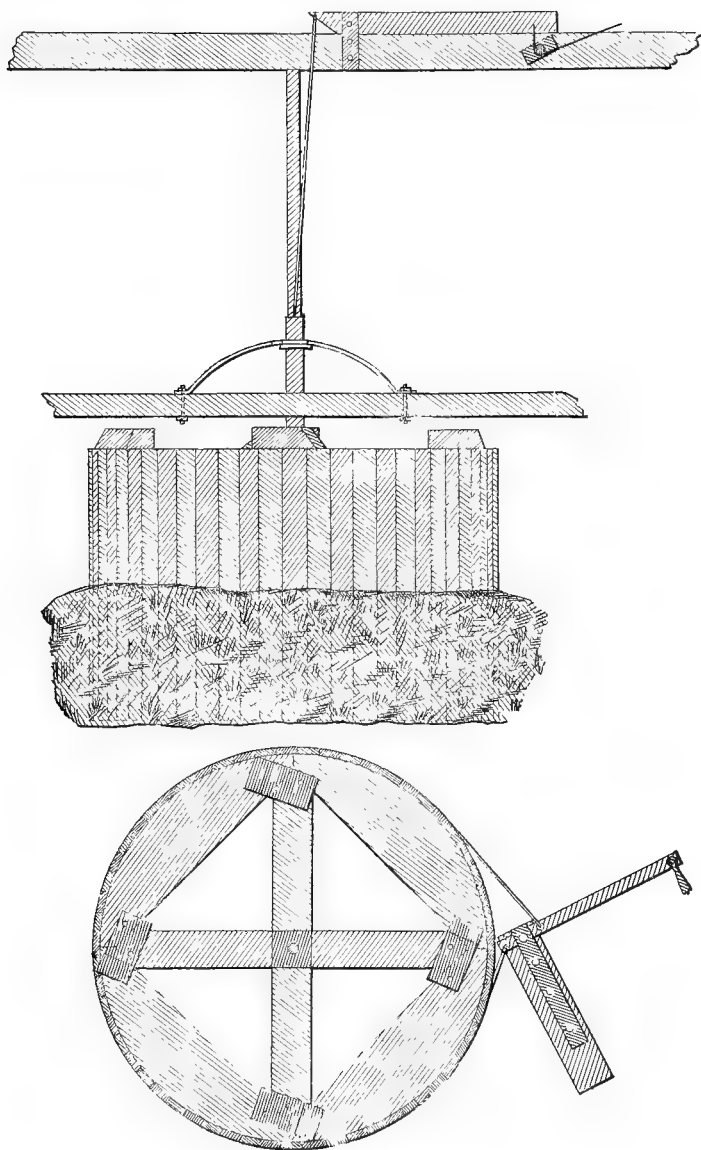


FIG. 17.

Each additional gearing involves a large percentage of loss, and there is little to spare from the average man-power of 5300 ft. lbs. per minute. They are, however, useful for incidental purposes in handling heavy pieces of timber, machinery, and pump pipes. A given power can only equal a certain product of weight and velocity ; for an increase of speed, the weight must be proportionately diminished. With a 15-inch crank-arm, 12 revolutions per minute coil 25 feet of rope on an 8-inch barrel, and with this speed of hoist the greatest load that may be moved under the circumstances by an average laborer is 214 lbs. Friction and stiffness of the rope will reduce this. A 150-lb. load can be raised at a speed of only 35 feet per minute.

To increase the efficiency of the windlass, the barrel may be lengthened and two buckets be suspended from the ends of the rope ; one descending balances the other ascending. The length of rope is a few coils only greater than the depth of the shaft. At the start the weight to be hoisted is only the contents of the tub plus the rope. This weight diminishes in rising till at the top it is the contents minus the rope. This does not, however, obviate the great stress from inertia which arises at the moment of starting. For this reason single or double conical barrels are used, on which rope is coiled in such manner that the empty bucket is hung from the larger diameter, while the rope from the loaded tub at the bottom is wrapped around the smaller end. The tubs balance each other, but the empty acts with a greater leverage than the loaded tub, and thus assists the power in overcoming the inertia. After the hoisting is under way, the empty and its lengthening rope uncoils with diminishing leverage, while the load with its shortening rope gradually winds on a larger diameter of the cone. The buckets do not meet in the middle of the shaft, but in the middle of the number of revolutions of the barrel. In any event the pitch of the cone must be calculated for the given conditions of depth and load, otherwise its advantage is manifest only at the start. When properly constructed, the conical drum is to be recommended. Not many attempts are made to apply this mechanism to hand hoisting, but it is rapidly com-

ing into favor with horse and engine power, notwithstanding that it is dearer than cylindrical drums.

14. Manual labor is manifestly too expensive to be regarded as any but a temporary expedient for hoisting.

When the height of the hoist is over 60 feet or the output more than 5 tons per shift, horse-power is employed to advantage. The average horse develops an effort of 135 lbs. when walking at a speed of 180 feet per minute. This is much below the theoretical value fixed for a horse-power, yet it represents the results of many tests upon the energy of the animal, which will raise 8 tons 200 feet per day.

For greater depths or quantity two horses are used. But when a still larger quantity is to be handled, a more efficient and economic power is employed. The utilization of horse-power is but an intermediate step in the history of many mines. Where water is scarce, fuel dear, and the transportation of machinery difficult, the horse-whim serves temporary ends as a simple and tolerably satisfactory hoister.

The invariable arrangement is a wheel-and-axle machine, consisting of a drum and driving-beam to which the horse is harnessed. Two sticks 6×6 and 9 feet long are mortised together at right angles to each other, with four 4-inch planks trimmed to the quadrant of a circle. These are held a foot or two apart by studs, and to them 3-inch plank staves are spiked to form the barrel, which, though upheld on the axle, turns freely about it. The axle is a round 2-inch rod stepped in a block of stone, and held at the top in an iron socket on the span-beam. The latter is 10 inches square, 36 feet long, supported on legs mortised and strapped to it. A square iron axle-rod fastened to the drum and turning with it is often seen, but is not so good as the free axle. The entire frame can be built for \$100. The drum may be above or below the driving-beam.

A derrick frame is necessary over the shaft at a height sufficient for convenience of handling the hoisted tub. The hoist-rope passes over a sheave at the top—direct to the drum if set up high over the driving-pole, or under another pulley at its

base if the drum is close to the ground. The latter arrangement is cheaper to build, but is wasteful of power, particularly so with a wire rope. For lowering the tub, a lever is in reach of the driver, by which the driving-beam may be disengaged from the drum and the tub lowered by its own weight, uncoiling the rope from the drum. A band-brake $3 \times \frac{1}{4}$ regulates the speed. The brake must be set so as to work with the motion, not opposite to it; and the brake force exerted to produce larger tension in the driving, not in the slack, portion of the band. The lengths of the driving-beam and the diameter of the drum may be altered at will, but the ratio between them is also the ratio of the speed of the horse to that of the hoist. (See Fig. 17.)

If circumstances permit, two ropes may be operated from the same drum, one ascending, and the other descending. Conical drums may also be employed. Iron-framed whims are on sale, whereby a drum is horizontal and turned by a bevel-gear on its axle, fitting to another at the central end of the drive beam. While convenient and easy to erect, the introduction of the bevel gear involves additional friction.

The plane of the derrick pulleys should be tangent to the drum, and the latter far enough away that the rope may coil and unwind freely without chafing on its adjoining coil. This is accomplished by making the point of departure of the rope from the pulley at the same height as the central coil. Where the full and empty tubs are simultaneously operated, this can only approximately be attained. If greater nicety is desired, the pulley slides on an inclined plane as the coiling or uncoiling proceeds, or else the drum shifts its position by turning on a screw-thread. No lateral motion is allowed to the sheave, since the rope must occupy a central position in its own hoisting compartment.

15. When the engineer is compelled to plan for greater output, he is met with many difficult questions. The probable conditions, as well as the actual output, must be the guide, and common prudence suggests that the machinery be in excess of the immediate wants, which are more urgent with the cheapness of the mineral. Undoubtedly, permanent hoisting

machinery must be placed sooner or later; though the sooner the better, it may momentarily be a question of available capital. With rich mineral and small product, the use of the horse-whim may be justified; yet money is wasted with each day. A 28-horse-power engine can do as much work as, at less cost than, 300 men on a windlass, or 35 horses on whims.

In selecting boilers, calorific economy, the amount of repairs, and the duration are to be compared. The records of the several types are easily had, and the facts gathered. The return tubular, or portable, boiler is being superseded rapidly by the multitubular, which in turn has a most active rival in the safety boiler. The efficiencies of the types average about 50, 60, and 90%. The relative initial cost at Denver, in place, is not far from 1-1.4-2.5. The last-named boiler consists of a series of connected tubes carrying the water, and whose outer surfaces are in contact with the fire. They are also called sectional boilers, are handily transported, safe, quick to clean, require little repairs, give dry steam and a high saving in fuel; particularly is this the case with high-pressure steam.

The tubular connections of the Babcock & Wilcox and the Root sectional boilers are illustrated in Figs. 18 and 19. Boilers should be set up as high as is consistent with the fireman's duties. An excess of boiler power is advisable for emergency.

The standard boiler horse-power, as adopted by the American Society of Civil Engineers, is the hourly evaporation of 3.55 gallons of feed-water at 100° Fahrenheit to 70 lbs. gauge pressure (33305 British thermal units per hour.) A storage of an ample supply of the purest obtainable water is urged. Incrustations of solid matter deposited from the water are frequent causes of explosion, in consequence of the destruction caused by the contact of the scale, as well as by the overheating necessary to evaporate the water. The presence of a scale one-sixteenth of an inch thick on the tube causes a loss of 13% of the fuel heat. Filtration through hay in the bottom of a tank will remove the muddiness; but the solids in solution are rendered innocuous only by the use of alkalies. Common

washing-soda is as effective as any, and much more so than the majority, of the nostrums sold as "anti-incrustators." A little glycerine—one pound to every 400 gallons of water—is beneficial. An excessive use of any of these causes "foaming." Tannates or organic compounds are injurious. Frequent "blowing off" and cleaning every week is necessary with the common tubular boilers.

The consumption of fuel per hour averages 3.5 lbs. per horse-power with the common boiler and ordinary engine. A cord of well-dried spruce is capable of running a common slide-valve hoister 320 hourly horse-powers; a ton of lignite, 470; a ton of anthracite, 650; and a barrel of petroleum, 170. Thus, a barrel of oil is worth more than 0.5 cord of wood or 720 lbs. of lignite. Firing with a thin bed of coals, and a proper lubrication of the engine, will reduce the quantity of coal consumed materially. The firing of boilers should receive attention to prevent a waste of the evaporating power of the fuel. While every possible plan is adopted to increase the economy of the engine-working by the use of fine machinery and lubricants, the boiler, which is the power-producer, is neglected. A proper relationship of fuel to grate-surface is essential. The fuel should be spread in an even, thin layer over the grate and consumed at an hourly rate of not over 20 pounds per square foot of grate. If the grate is too large, the temperature of combustion is too low. High temperature in the furnace is the first condition of economy, the second being that the heat produced shall be absorbed as completely as possible by plenty of clean heating-surface. A grate-surface too small causes clinking of the coal.

The greatest waste of heat is due to the improper supply of air. Experiment has shown that bituminous coal requires 150 cubic feet of air per pound. Less air than this results in smoke, an excess chills the coal and carries heat up the chimney. A dull sluggish fire would suggest a combustion which could be remedied by better draught. Bituminous coal usually requires an auxiliary gas-producing oven to give economic results and save the heat otherwise lost in the smoke of volatile gases.

A boiler should evaporate 15 lbs. of water per pound of good clean coal. But it would be safe to say that 75 per cent of the boilers in use evaporate not over $10\frac{1}{2}$ lbs., while many mine boilers show only 4 to 6 lbs. of water boiled away by the combustion of a pound of average coal. The results from boilers depends more on the character of firing than on any other element. Frequent stirring at proper times, cleaning of the flues every week, and the purification of the feed-waters are essential elements of economical combustion.

The feed-water should be purified in tanks, not in the boiler, to secure the best results. The sediment is then taken out of the water in the heater, which should be large enough to allow plenty of time for settling before the current can carry the impurities into the boiler.

The McClave and Howe grates have rendered it possible to utilize fine sizes of coal, and mechanical stokers have proven their ability to economize in fuel to a high degree. A steaming-coal should kindle readily, and burn steadily without clinkers. A superior coal is that high in fixed carbon; a little volatile matter will assist the firing. Slack is nearly equal to coal in calorific value, but it has, usually, too much refuse to be acceptable, except where the transportation is cheap. On account of the variant "personal equations" of coals, experiment alone will determine the best coal for a given grate.

A great saving in fuel can be had by heating the feed-water before injection. A braced tank of matched, dovetailed boards receives the exhaust steam and well serves the purpose of a heater. The open end of the exhaust-pipe blows steam over the surface of the water, and is placed higher than the waste-pipe which maintains the level. The exhaust-pipe may also be laid as a coil, with its end discharging into the air. This is better: otherwise the oil which had previously been used for lubrication gives trouble when carried over into the exhaust. A wire-mesh strainer proves a fairly effective device for catching the oil if it is placed on the level of the discharge. The "economizers," or combination heaters and purifiers, are recommended. In these, some of the heat from

the chimney-gases is employed to heat the feed-water (Fig. 18). The benefit from them increases with the wastefulness of the boiler. There is little gain in adding them to a condenser. The purification of the feed-water should be performed elsewhere than in the boiler. The practice of piling coal in the open air and exposing it to the weather is deprecated. Not only is the quality of the coal injured, but spontaneous ignition is liable to take place if the weathering is prolonged. Coal condenses gases within its pores and over its surface in proportion to the area exposed, and hence to its fineness. Freshly-mined coal absorbs fully three times its volume of gas, and this mechanical action results in the production of heat and the combustion of the fuel. This increases with the rise of temperature and with the exposure to alternations of damp and dry air. Fuel should therefore be housed, greater attention being given to the small sizes. The inflammability of anthracite is less than that of gas-coals, and they, in turn, ignite at a lower temperature than lignite.

It was believed that the presence of pyrites in the coal was the principal cause of spontaneous ignition, but it is now conceded that the presence of moisture in addition to the absorption of oxygen by the coal induces a rapid deterioration that may result in fire. The only remedy against ignition is in a continuously free circulation of air—a condition almost impossible to attain. Wetting the coals at first stops action, but in a short while it hastens the process which the presence of fine coal facilitates.

Frequent analyses, by the d'Orsat apparatus, of the chimney gases will determine if the air is being completely and properly employed in the combustion of the coal.

The installation of the machine plant in a suitable place, of ample size and within as small a compass as possible, is a matter of great moment to the engineer. And it may here be remarked that while the finest and best machinery is in the long run eminently economical, the question of finances and capital is not to be ignored. Until the mine has been developed, the engineer should content himself with expedients. A large boiler and

engine is more efficient in the development of power, and with less attendance, than a number of small plants. A distribution of its power to remote points by electricity, compressed air, wire-rope, or string-rops, is possible with little loss; and few arguments favor the establishment of numerous small outfits, so often encountered in extensive mines, except those which

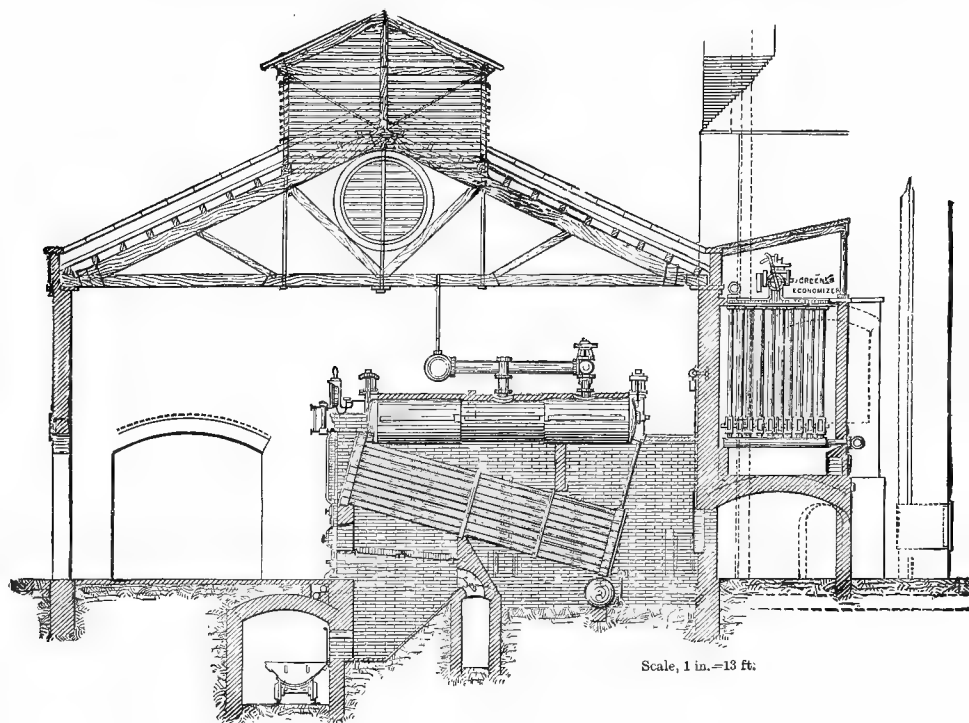


FIG. 18.

also excuse the lack of a system. Fireproof housing is indispensable; not only because of the weather protection given the plant, but also because injury to the motors threatens the dependent machinery, affecting the safety of the men and the security of the mine. Many causes of fire above and below ground contribute to make this essential. The buildings should be as far away from the shaft as is consistent with

economic work. But two reasons exist for having hoisting-appliances close to the shaft,—quick landing of the cage, and dumping facilities. The former is secured by isolation of the hoister-man, who has a free view of the shaft-mouth from a position high enough up and far enough away to give a good lead without allowing oscillation of the rope (Fig. 16). A complete

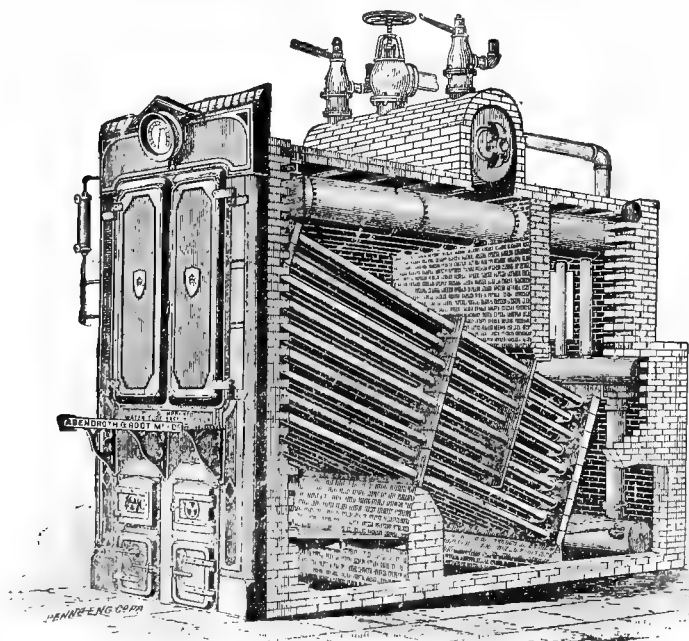


FIG. 19.

system of signals gives him control. Where the output is small, the tubman, who attends to the dumping, can also give the signals, or, by having brakes, levers, and throttles at hand, he may manage the engine from a distance. A gravity plane from the shaft to the mill or dump, under the charge of the surface unloader, will dispose of the ore and waste.

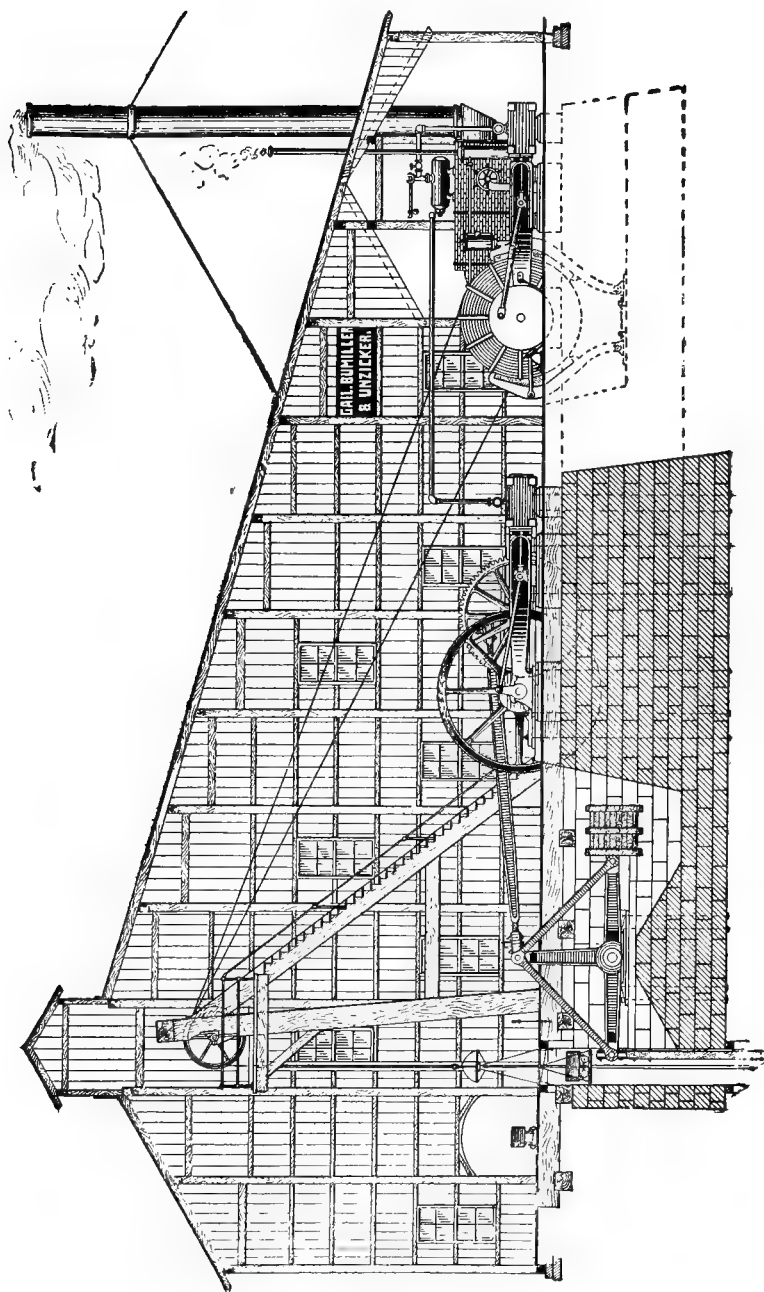
The hoister is best located opposite to one end of the shaft, with the boilers as close as possible, and the fan connected to its compartment, containing also the pump-pipes and ladder-way.

The foundation of the engine is prepared before its arrival by laying an accurately constructed template in the proper position for the engine, with the bolts standing erect from it, long enough to reach through the masonry to the bed-plate. The foundation of concrete and brick, gives a perfect bearing throughout for the bed-plate. Timber seats are often used, but are too elastic for rapid hoisting, which demands rigidity. Only second- and third-motion haulage-engines may have timber foundations because of their uniform and slow motion.

A hoister may be described as a simple steam-engine attached to a drum for receiving the rope. The combinations are numerous, from that of the upright boiler and its engine on the same base, to the vertical beam condensing-engine connected with the drums on separate foundations. The latter is of European preference; the former for prospecting-service only. As a rule, however, the cylinders are horizontal.

The intermittent operation of hoisting an un-uniform load at high velocity is not inherently economic. The regulating fly-wheel had long since to be abandoned as dangerous for reversing engines, so all the means of conserving power must be by the use of condensers, high-pressure and expansive working. The comparatively high initial cost of these improvements constitutes the objection to their more extensive use, yet this bears a small ratio to the saving in fuel. Ordinarily, 33 per cent. is quoted as the fuel gain in favor of condensers where plenty of water is procurable. The exhaust steam is condensed in a water-jacket reservoir, and pure distilled water obtained for return to the boiler. The best remedy for overloaded engines is to add a condenser which, for ordinary pressures, will shorten the cut-off. It does not usually effect any saving if attached to a lightly loaded engine at high pressure. It is better to reduce the pressure.

16. The designs of hoister are numerous, but the types few (tail-rope and other engines are embraced in this discussion), divided into the slide-valve and cut-off classes. The slide-valve allows of the use of the steam at full pressure throughout the entire length of stroke, and simply acts to



reverse the direction of the induction. Balanced slide-valves are much admired. Portable and pony hoisters are of this class. The larger and more perfect types of engines work the steam expansively, by cut-off or by compound cylinder. The mechanism for shutting off the steam supply before the piston reaches the end of its stroke may be fixed or variable, automatic or adjustable.

A fixed cut-off is intended for a constant duty, as for marine purposes. In it the induction-valves are set in some such fixed position that the gearing, or governor, will always close them when the piston has reached a given point of its stroke; after which the piston is urged by the expanding steam. In the variable cut-off, only a single valve is used, but it operates on the exhaust as well as on the induction, and steam is caught on the exhaust side, and capacity is lost. The Meyer's cut-off is adjustable and can be set by hand, at any time, to regulate the point of the cutting off. The exhaust is not interfered with, because an auxiliary valve closes the induction. In the automatic cut-off, the governor operates a valve for the induction and one for the exhaust, each with a motion independent of the other. In one class of the automatic, the valves are inoperative until the engine has attained sufficient speed, when they are brought into play by the governor, and the speed becomes constant.

The Corliss is a fixed type in principle whereby a trip-gear, regulated by the governor manipulates the valve at each stroke. It is practically instantaneous, and is increasing in popularity among mining men. Its best results are at 0.2 cut-off with an average load, and at this rate generates about 40 per cent more power than is attained from an equal amount of fuel in the plain slide-valve. A 100-horse-power Corliss will save fully 300 tons of coal annually. (Figs. 21 and 27.)

Still greater economy in generating power is had by a high degree of expansion and the tendency of the times is toward this goal. The final pressure in a non-condensing cylinder cannot be below 19 pounds, as it requires three or four pounds to expel the exhaust and to drive the engine. A short cut-off,

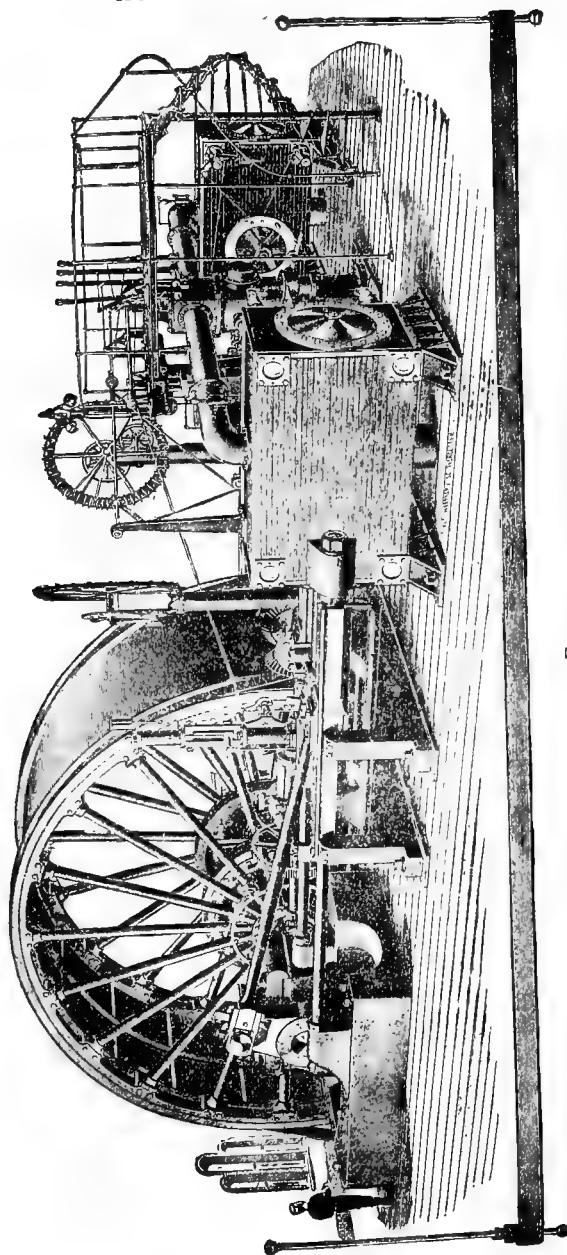


FIG. 21.

therefore, necessitates a heavy boiler pressure. Again, the rate of expansion is limited because of the condensation accompanying steam highly expanded. The ratio, or the minimum pressure, is also prescribed by the conditions of the hoister. It is expected to be able to start its load from any point in the shaft and at any point of its stroke. With a low terminal pressure a fast hoist may not be possible unless the cylinder be large. So, valve-gear cannot be set permanently for a short cut-off without a high initial pressure. Moreover, a high rate of expansion means a low average pressure throughout the stroke, and this involves the use of a large cylinder for a given power. So that, while hoisting is medium slow, the ordinary pressure is permissible; but, for reasons suggested, only high boiler-pressure and considerable expansion can satisfy the tendency toward rapid winding. This is creating a demand for high-class machinery, the which is being rapidly installed, heavier and larger than formerly.

The evils attendant upon high single expansion are best remedied by the use of compound cylinders. The steam, after doing duty in one cylinder, is discharged into a second larger one (sometimes from there into a third), where it is expanded before being exhausted into the atmosphere or a condenser. By this means each cylinder has a range of temperature and pressure which is not large, but previously fixed. Triple—and even quadruple—cylinders, and expansion, are employed; but their advantage is developed only by a high pressure, a comparatively steady load, and heavy work; hence not for hoisters. The cylinders are placed tandem or alongside, the pistons being on the same rod, or united by cross-head; each piston furnishes an independent force, and the power of the engine is equal to the sum of the pressures operating. The steam-cylinders should always be enclosed in a poor-conducting envelope or else by a steam-jacket. This prevents condensation and allows of a higher rate of expansion, saving fully 8 per cent of fuel.

The amount of steam consumed per hourly horse-power is a very variable quantity, depending upon the type of en-

gine and the internal friction. Condensation in the supply-pipes, or in the cylinder, wastes considerable steam; an under-loaded engine wastes almost as much as an overloaded engine, which would exhaust at too high a terminal pressure. A common throttling engine consumes about 45 lbs. of steam for every hourly indicated horse-power; a fair automatic cut-off engine, 35; a compound, high-speed engine, 25; while the highest type of double-expansion consumes 14 lbs. Even the latter represents but one-seventh of the heating value of the fuel. On this basis, all consumption of steam above 14 lbs. per hourly horse-power is preventable waste. Assuming that one pound of water requires 1100 heat-units for its evaporation, then 14 lbs. require 15,400 heat-units. As one horse-power per hour is equivalent to 2569 heat-units, then the efficiency of an engine compared to its fuel is 16.6 per cent, or 10 per cent if the boiler has a fair efficiency of 60 per cent.

Many boilers show a high apparent evaporation in consequence of furnishing "wet steam," while practically they are anything but economical. Water carried over into the cylinder entrained in the steam, having no useful effect, is indeed a detriment by its collection in the cylinder with consequent injurious results. The cause of this "priming" is either impure water, too much water, or the foaming produced by some forms of scale-preventers. The amount of priming in different boilers varies greatly, tests showing as high as 18.5 per cent of water in the steam. Steam carrying but 3 per cent may be termed "dry." The use of water separators and a subsequent superheating not only reduces the amount of priming but increases the efficiency of the plant.

Steam-pipes should be covered with a non-conducting material. A pipe 4 inches diameter, 86 feet long, without covering loses one horse-power. An inch of covering increases the efficiency, one horse-power being lost for 284 feet of pipe. The loss of heat and of power varies directly with the increased surface which is exposed, and inversely as the square of the thickness of covering.

It more frequently happens, however, that economy of fuel is the last consideration, the other requisites being more

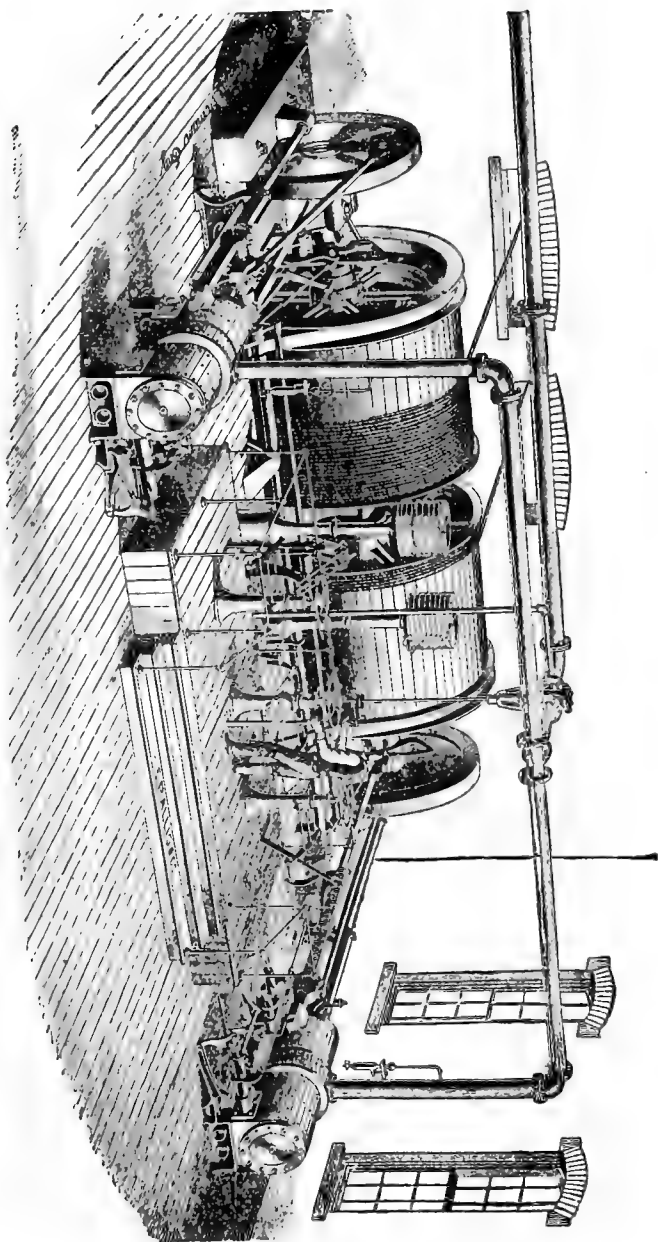


FIG. 22.

urgent,—simplicity and safety, for example. The engine must be under complete control, and capable of quickly attaining full speed.

The communication of motion from engine to drum may be direct or secondary, and the mechanism reversible or not. In the former class the load is hoisted, held, and lowered by steam, the reversible movement being easily imparted at will. It is perfectly safe and speedy, but does not admit of expansion-gearing. The piston is directly coupled to the drum, and this is the only form by which high speed can be quickly attained (Figs. 21, 23, 24). Pedestals for supporting the drum-shaft are upright or only slightly inclined.

Second-motion engines include all those by which the power is transmitted to the drum through a driving-shaft carrying some form of gearing or friction wheels, or through some variety of friction surface. Its introduction is intended for the regulation of the speed; but gearing involves a loss of power, so that, where its intervention could be dispensed with, it would be highly desirable so to do. See illustrations, Figs. 22, 27, 28, and 29.

The driving-shaft may carry 1 or 2 pinions which mesh with an equal number of spur-wheels keyed with the drum to the shaft. The ratio of this gear is between 4 to 1 and 7 to 1. Some quarry engines are provided with a double set of gear for rapid and slow motion (Fig. 22). The ratio must be the same as that of the weight or speed of the load to the capacity of the cylinders. Power and speed are convertible terms. Occasionally, instead of spur-wheels, V friction wheels are encountered. The small wheels or pinions are in continual revolution, a lever throws the drum against them, and motion is effected. Great friction is thus obtained without excessive pressure; but they soon fail to work properly, because of the irregular wear of the grooves.

For pony hoisters, friction wheels are used instead of the V-grooves. The revolution of the drum is obtained by bearing it against a *papier-maché* wheel on the driving-shaft (Fig. 30). Only for prospecting-work are they advised. The sur-

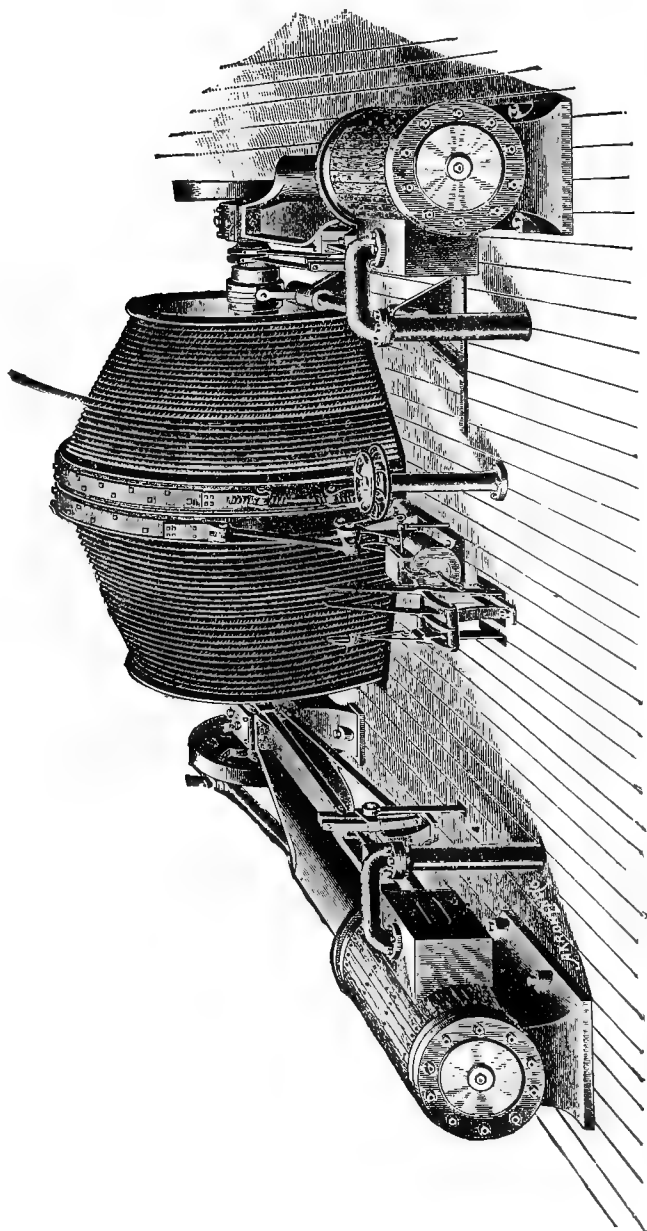


FIG. 23.

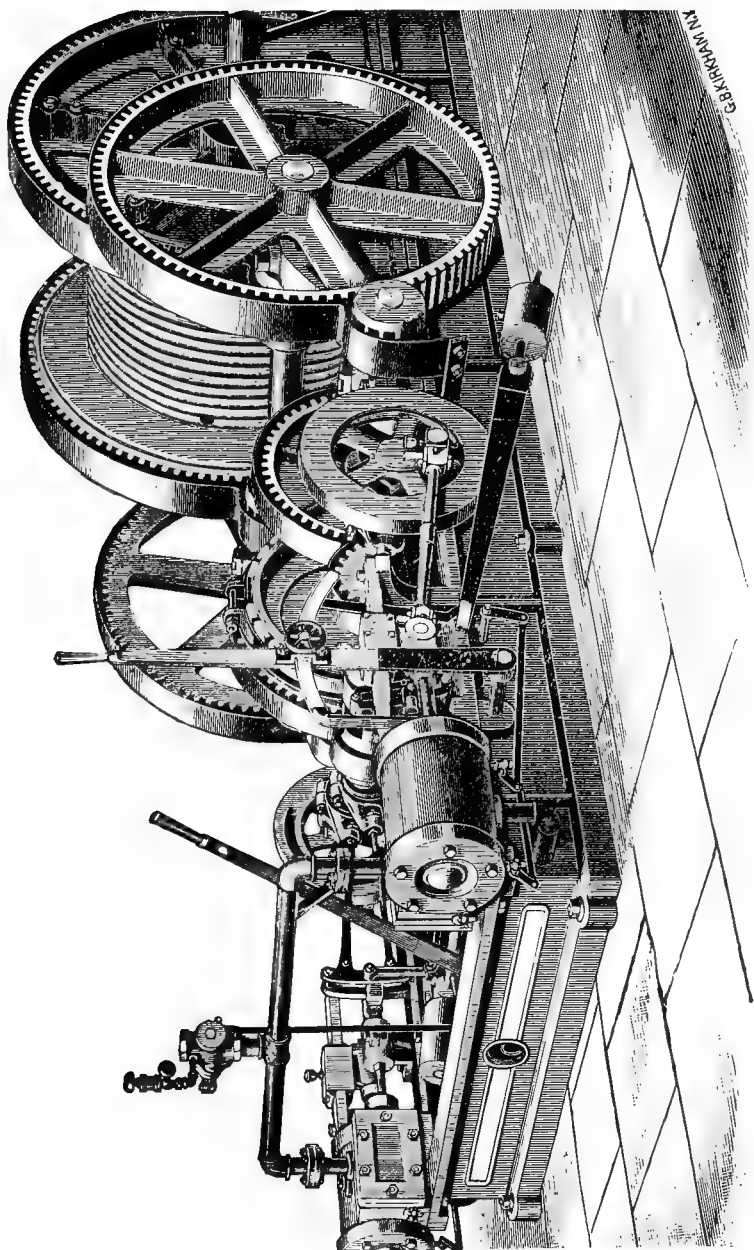


FIG. 24.

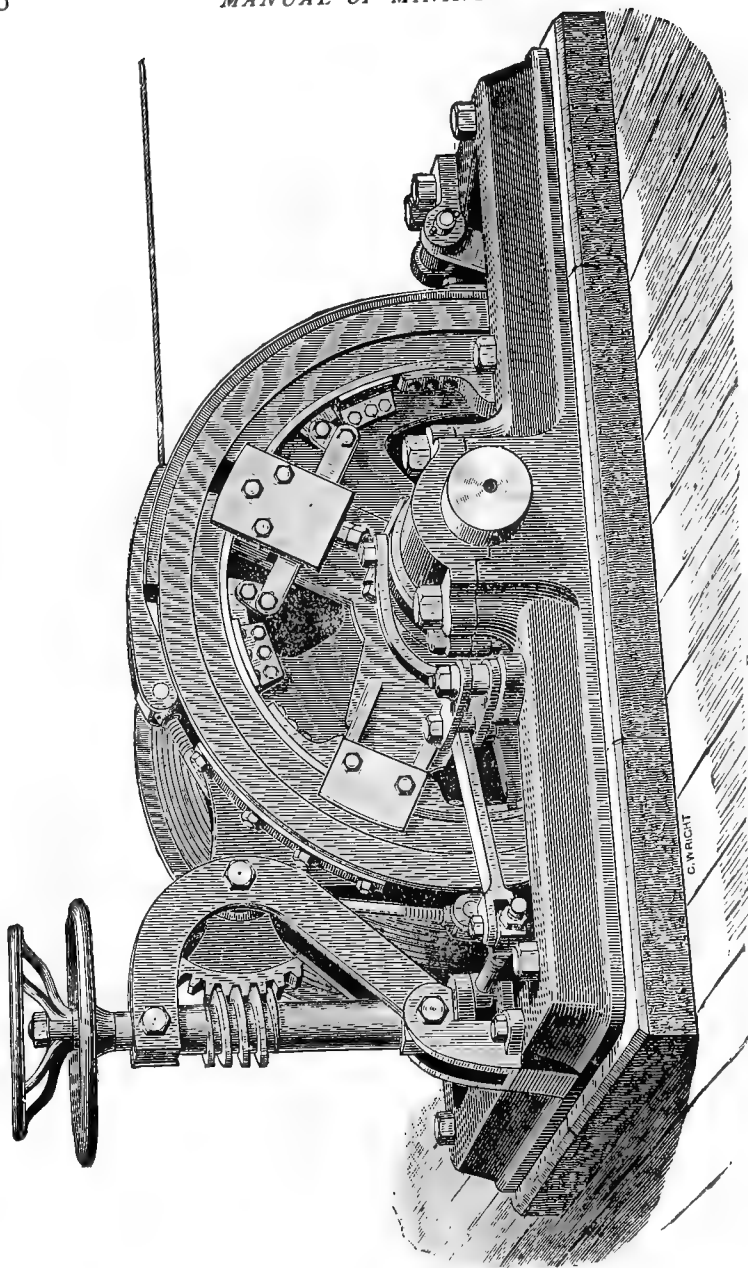


FIG. 25.

faces must be kept dry and free from water, steam, or rosin. A word of caution against the use of rosin, coal-tar, etc., on friction surfaces. Undoubtedly they increase the friction momentarily, and the men are prone to use it, but the heat developed soon melts or gums it, and the surface has become too slickery for remedy by other means than a cold chisel and elbow-grease. The author has posted, about his machinery, notices of prohibition against the use of these materials.

17. The commonest class of hoisters, after the direct-connection engines, is represented in the types of friction-clutches engaging the drum in various ways. They are quick, powerful, noiseless, and give little shock. One form of internal friction consists of driving-band and four arms keyed to the drum-shaft (Fig. 25), the gear-wheel of which receives the motion from its pinion on the driving-shaft. The drum is well bushed, free on its shaft, and turns as soon as the band is expanded to contact with its inside rim. This the engineer manipulates by a lever. In another variety the contrivance is a set of curved friction-blocks on spokes fast to the shaft. These bear against a similar surface inside of the drum, when the latter is slid up to the engagement. The reverse is also in use, whereby the friction-blocks are forced into the drum. The device resembles a dished-wheel. Internal gearing regulates the hoisting-speed without altering the engine-speed, and is quite successful. Of the external frictions we have a sectional driving-band attached to a two-armed driver keyed to the drum-shaft and operated by sliding-blocks and bell-crank levers. When the sliding collar is moved towards the drum, the band clasps it and imparts the motion (Fig. 26).

For variable lengths of hoist, drums have an inside pinion gearing into a rack on the drum. This can be slipped out at will and the free drum pays out rope until the proper level is reached. Then the pinion is returned, and the drum is ready.

These appliances are keyed, by the troublesome feather, to round shafts. Octagonal shafts are preferable, though more costly. Some hoisters are provided with both link-motion for men, and frictional-clutch for other purposes. The best type

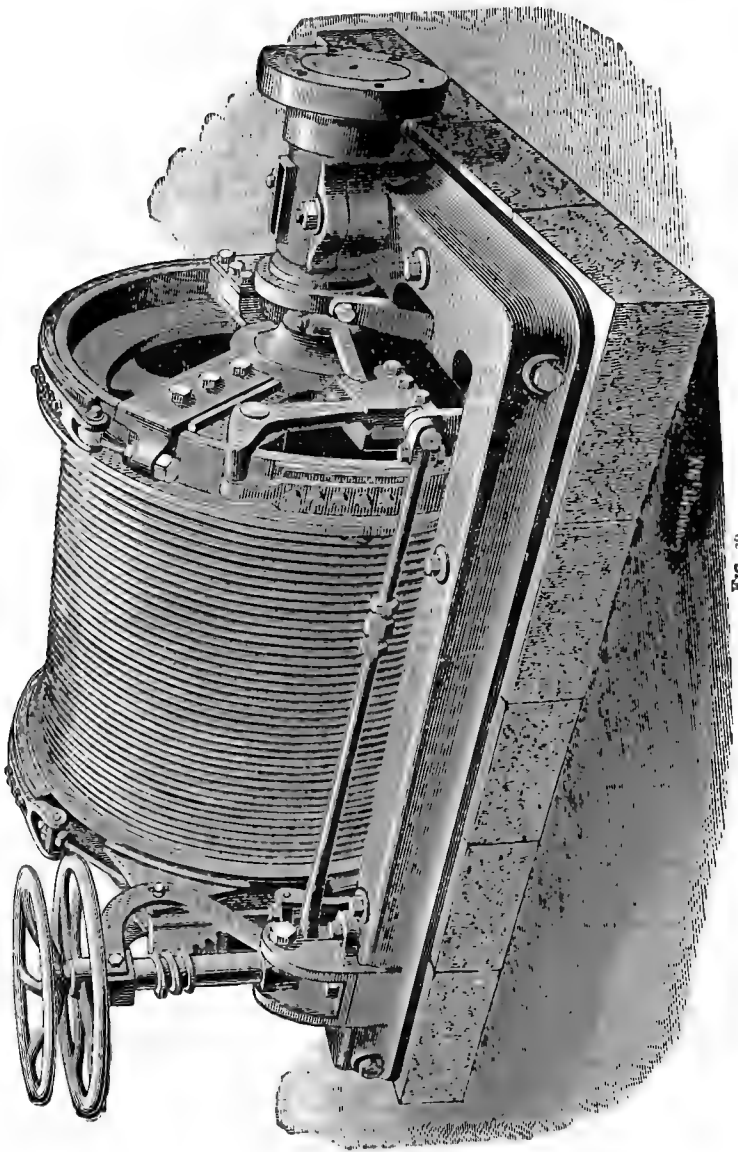


FIG. 20.

is a double-cylinder, high-pressure engine fitted with variable expansion and reversing-gear, having pistons directly connected on the crank-arm shaft, on which are the drums with steam-brakes. In some localities, the drum, on a separate foundation, receives its motion by wire rope, electricity, or otherwise. To avoid dead-points, the drum, or its driving-shaft, is coupled to two independent engines by cranks at right angles (Figs. 24 and 27). This arrangement is more equable and safer than the single cylinder.

The direct-connection hoister and the reversible-link hoister, Lidgerwood pattern, Figs. 24 and 28, are always in gear, have reversible valve-motion, enabling them to turn "over" or "under." They employ steam, whether for hoisting or for lowering, and are perfectly safe for carrying men. By varying the position of the cut-off, the speed is controlled. With the other classes, hoisting cannot begin until the drum be thrown into gear after the engine has "got up" speed. Moreover, their motion is positive and continuous, non-reversible. To lower the load, the drum is disengaged from its gear or friction, the speed of the uncoiling rope regulated by a brake.

The brake is either an iron band claspings the rim of the drum, blocks of hard wood on end against it (Fig. 29), or V-grooved wheels. The first is the least troublesome and is safer, though having less friction than wooden blocks. The band-brakes are represented in Figs. 22, 23, and 24.

The brake is suitably applied by a simple lever, a band-wheel and worm-screw (Fig. 25), or an auxiliary steam-piston, according to the size of the hoister (Fig. 22). The last-named acts powerfully,—indeed, too suddenly,—causing injurious shocks, against which are numerous devices. The leviathan at the "Calumet and Hecla" mine requires a small auxiliary engine to stop and start it.

Each compartment of a hoist- or tram-way has a rope and a drum. Sometimes, for double-compartment shafts, two ropes are wound—one over, the other under—on the same drum, which is directly connected to a reversing-engine. Ordinarily, however, for multiple-hoisting, each rope has its own drum in-

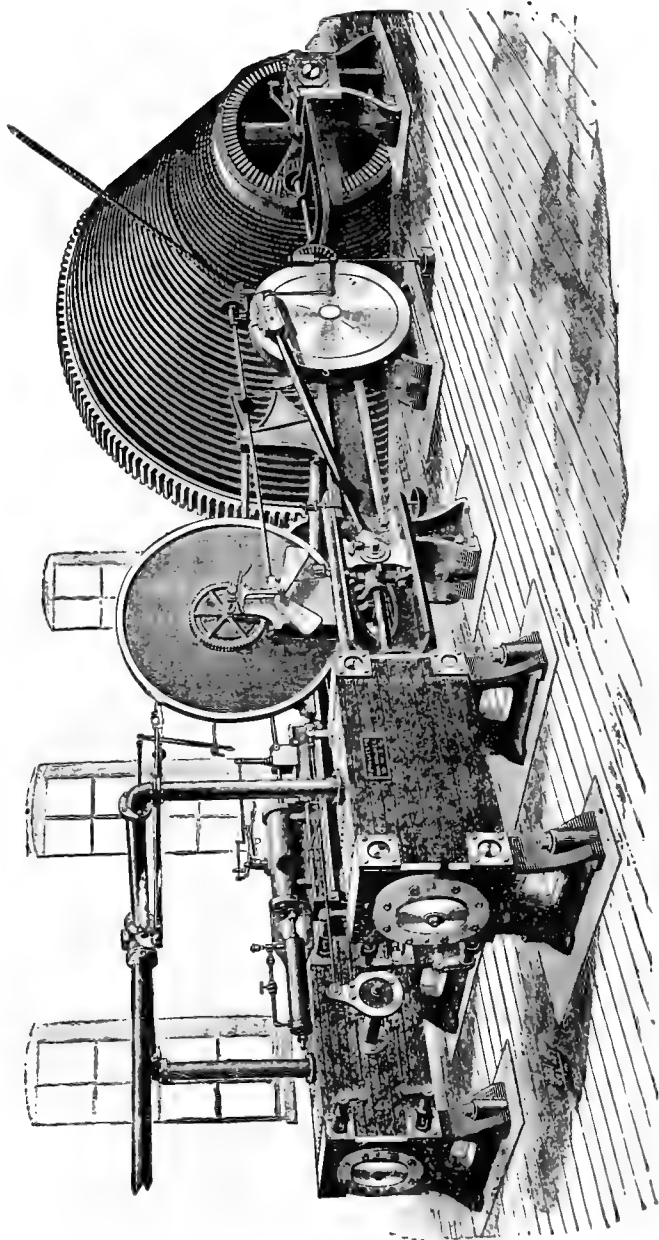


FIG. 27.

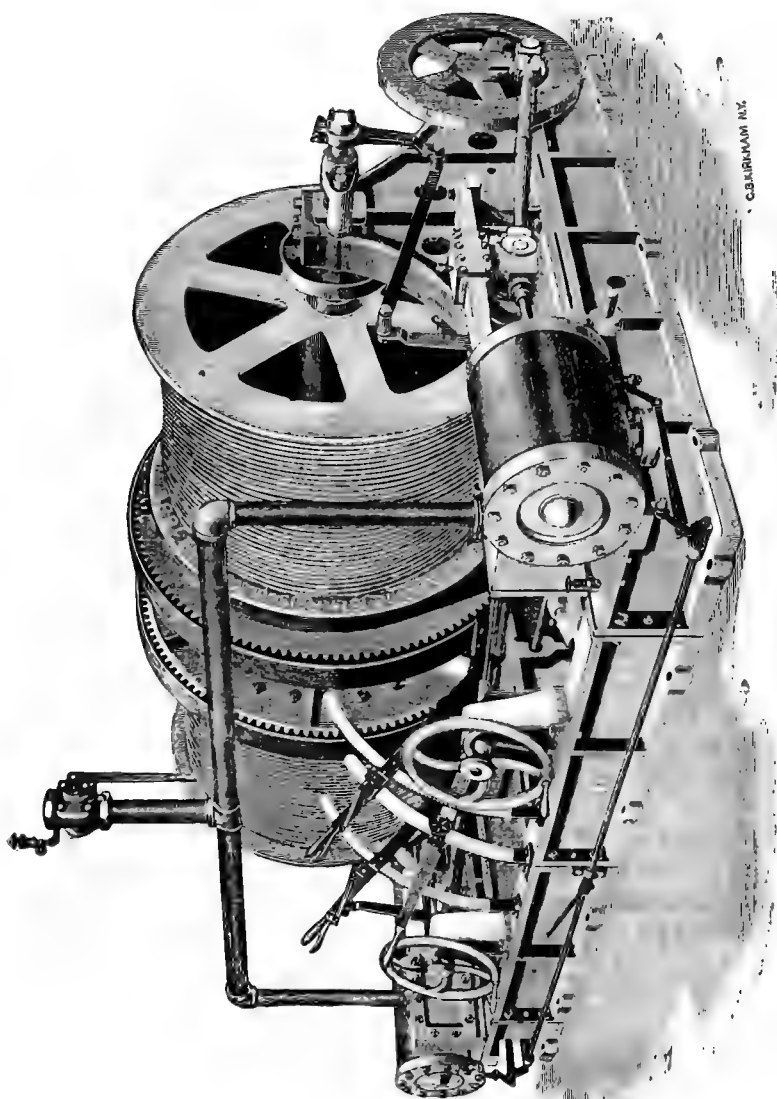


FIG. 28.

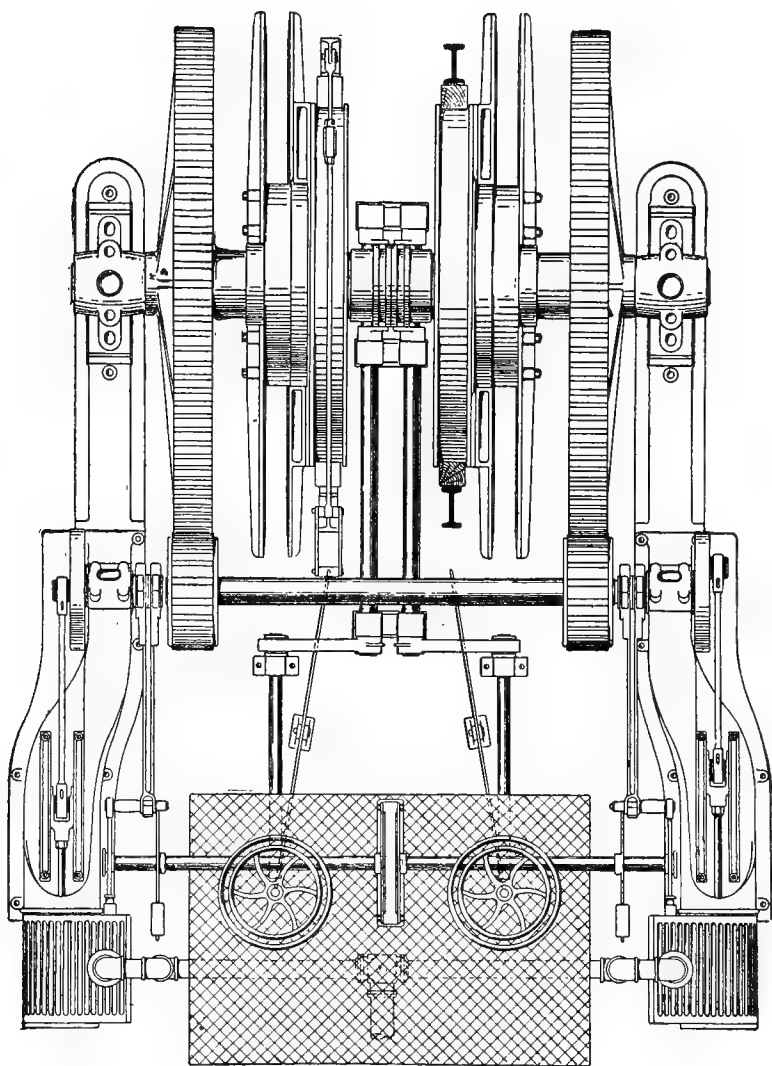


FIG. 29.

dependently controlled. The drums are all geared to the same driving-shaft and placed centrally between the steam-cylinders. The two cylinders may or may not be fed by the same throttle.

Drums up to 8 feet diameter are cast whole; above that, in segments, cylindrical or conical for round ropes, or in reels for flat ropes lined with plank to reduce the wear of the coils. The diameter of the drum is at least forty times that of the rope, and its length sufficient to accommodate the full rope length. Whether cylindrical or conical, the groove is turned spirally over its face to receive the entire length of rope (Figs. 27 and 28). Its diminishing pendent weight, as the hoist progresses, throws a markedly unequal work on the engine, and, without a variable expansion, regular speed cannot be maintained. Some approach to equality is established with two cages or tubs going in opposite directions, but that is only for certain moments (Figs. 20 and 22). Nor is the inequality of motion effectually remedied by flat ropes winding on reels (Fig. 29); the start is eased, that is all; the decrease in the load, due to the shortening rope, is not compensated for by the increased leverage. Equalization during each revolution is what is aimed at. Conical or tapering drums partially attain the desired end; tapering ropes diminish the difference between the initial and final loads; and some compensation is had from the use of the flat rope and reel; but as yet no more satisfactory solution has been reached than that by counterpoises.

The conical drum, or fusee, is built for a given condition of load, speed, and depth. It may be single or double (Figs. 27 and 28). Its minimum diameter is determined by the size of the rope (see 26), beyond which the winding is on a surface described by a curve such that the moments of the ascending and descending load are constant, and give perfect balance for all positions. One danger with tapering drums is the tendency of the rope to slip from its place, where the angle is over 30° , but the spiral groove largely prevents this. In a

shaft whose depth or point of hoist is fixed, the equality may be maintained; but for purposes of hoisting in shafts operating mines of steep pitch, a conical drum cannot do service as a balance for more than one certain depth. In such mines counterpoises must be resorted to with cylindrical drums.

Below is given a formula for the computation of conical drums, taken from the *Colliery Engineer*, and another for the fusee, from an article by E. N. Rogers read before the American Institute of Mining Engineers.

In designing a single or double conical drum, the following formula may be used:

Let P be the horizontal distance between the centres of two consecutive grooves, and p the vertical distance between them (P is usually one half an inch greater than the diameter of the rope R). The ratio $p \div P$ is the angle of inclination of the face of the drum from the horizontal. This angle is never over 30° . The diameter, R , of the rope of requisite strength and length in inches having been calculated, thence the smaller radius of the drum, r , in inches ($r = 40R$ to $100R$), and the length, D , of the rope, in inches, to be wound upon the drum, decided upon, the vertical pitch p and the horizontal pitch P being assumed, the number of grooves, n , and the larger diameter, d , in inches, may be found by the following process:

c = the small diameter plus the diameter of the rope multiplied by 3.1416 $= (r + R)3.1416$;

p = twice the vertical pitch multiplied by 3.1416;

w = the horizontal width of the drum between the first and last grooves. Then

$$n = \frac{\sqrt{(2c - p')^2 + 8p'D} - 2c + p'}{2p'}$$

and $d = r + 2p(n - 1)$,

and $W = P(n - 1)$.

Employing similar symbols, Mr. E. N. Rogers has promulgated a formula for the double-fusee drum, which shall give an equalization of load at all points throughout the

journey of the cage, in which the falling weight of the cage and rope on one end of the drum shall balance the rising weight of cage and rope on the other end, in any position. Let r be the small radius of the fusee in feet, and d of the large end in feet; B the weight of skip or cage, and R the weight per foot of rope. Then for the position when the whole rope is suspended from the small end, while the cage is hanging from the large end,

$$(RD + B)r = Bd = M;$$

after one revolution, $(RD + B - \text{the weight of one turn of rope on the small end})r$ must balance $(B + \text{the weight of one turn of rope on the large end})d$, and still equal the constant $Bd = M$. So, for each revolution throughout the journey, this balance should be equal to M . This is satisfied by the formula

$$\begin{aligned} \rho = r \\ v = 2\pi n = \frac{\sqrt{\frac{M^2}{R^2} - r^4}}{2r^2} - \frac{\sqrt{\frac{M^2}{R^2} - d^4}}{2d^2} \\ \rho = d \end{aligned} + \frac{1}{2} \sin^{-1} \frac{Rr^2}{M} - \frac{1}{2} \sin^{-1} \frac{Rd^2}{M}.$$

The last two terms may be neglected without sensible error. In the above equation, v is the total arc of revolution described by the point on the rope at the small end between the beginning and the end of the hoist.

The curve of the drum is then constructed by substituting different values for r and d , which are the limits of ρ , placing the second member equal to $2\pi n$, and solving for n , the number of grooves. The various assumed values for ρ are radii of the curve at the various points along the axis which are at a distance from the initial point equal to horizontal pitch, P , multiplied by n . The curve so plotted is the section of the drum which will fulfil the conditions, furnishing an equalization throughout the journey, so long as the length and weight of the full rope remain the same.

In an example cited by the author, $D = 2000$ feet, $R = 3$ pounds, $B = 4000$ pounds, $r = 4$ feet; then $M = 40,000$ foot-pounds, and d becomes 10 feet. Solving the equation

and equating with $2\pi n$, we have for $d = 10$, $a = 4$, $n = 55.8$. That is, between the initial and final points, there are 55.8 revolutions of the drum. For $r = 4$ and $d = 9$, $n = 53.3$ revolutions; in like manner for $d = 5$, $n = 35.1$; for $d = 7$, $n = 44.4$; and for $d = 8$, $n = 49.5$.

Another form of counterpoise is a tail-rope extending from underneath one cage, under a sheave at the bottom of the shaft and up to the floor of the other cage. The dead weight on each rope is constant; the oscillation of the cage is reduced, a regular speed quickly attained; friction, also the size of the rope, is increased. A shaft free from impediments is necessary, as also a dry pit. Another plan employs a heavy weight, and a chain wound on an auxiliary drum in such a manner as to always balance the ropes in any position of the cages. On the drum-shaft a third drum is attached. This drum winds and unwinds a wire-rope, which with a chain at the end hangs down an auxiliary shaft, or down the ladder-way of the hoisting-shaft. The length of rope is such that when the two cages in the hoistways are passing opposite each other the entire chain is coiled up in a box provided at the proper point. The weight of the chain must be sufficient to balance the weight of the full length of hoisting-rope, and at all points in the hoist the amount of suspended chain must balance the difference between the weights of the two pendent hoist-ropes. As the loaded cage rises from the bottom the third drum commences to lower the chain into the box, and lessens the weight hanging from it to assist the engine in balancing the heavy rope being hoisted. When the cages pass each other the two ropes balance, and the chain is not operating, the counterbalancing rope being all paid out. See Fig. 240.

As the loaded cage continues to rise the drum commence to wind the counterbalancing rope in the opposite direction, thus raising the chain and bringing its weight into play as a counterpoise to the weight of the descending rope.

In a shaft of 2200 feet deep, the counterbalance-rope is 700 feet long and its chain, 580 feet, weighs 4320 pounds. The size of the chain is graduated to meet the varying weights

to be lifted. Thus 104 feet are of $\frac{5}{8}$ -inch chain, 162 feet of $\frac{3}{4}$, and 314 feet of $\frac{7}{8}$. The counterbalance may be so adjusted as to enable it to hoist from any level. Its saving in fuel is very great. It is simple and cheap, and works smoothly.

The Koepe system of winding meets almost all the requirements of a perfect equalization, and is highly efficient. It ensures against overwinding, decreases wear, and dispenses with the enormously heavy drum, using a sheave instead. The two cages are connected by tail-rope below and by main rope above. Then the engine does a steady, uniform work of lifting the net load only. Hoisting is possible only when the friction caused by the loads on both ends of the main rope is greater than the weight of the net load carried on the rising cage; and in several localities it has been abandoned, because, immediately after oiling, the rope would slip and the work was unsatisfactory. By counterbalancing, the work is only that of raising the live load + friction. Single hoists (unbalanced) are excessively wasteful in power and fuel, and hard on the brakes. The fuel value of hoisting 1000 feet of rope and a heavy cage thirty times an hour is no small quantity.

It may be well here to mention a most useful piece of apparatus, and by no means superfluous about a mine, for handling heavy articles—a portable tripod and a Weston differential-pulley block. It is simple enough to be manipulated by any one without fear of injury by rough treatment, and is exceedingly powerful (Fig. 31).

The following may serve as references for those desiring to further investigate the subject-matter of this chapter:

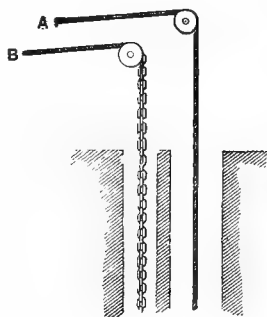


FIG. 30.

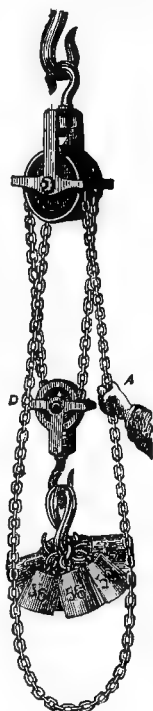


FIG. 31.

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· CHAPTER V.

ELECTRICITY AND WATER-POWER.

18. Application of electricity and water-power to long-distance transmission; comparison with mechanical means; universality, to all operations of mining. 19. Conducting wires, size, etc.; two-wire and three-wire systems; safe voltage; explanation of the electric units, and formulæ; conversion of electric into kinetic energy by motors; efficiency of motors; storage batteries. 20. Mode of obtaining water-power by the use of Leffel, Knight, and Pelton wheels; description, efficiency, and application of the plants and machines.

18. THE most valuable acquisition made to any branch of industry during the past few years was electricity, and with phenomenal rapidity it has gained favor. Not more than sixteen years ago electricity was a mystical force that was not suspected as capable of operating even a telephone. To-day the installation of a plant ceases to be a novelty; and its utility as an illuminator, and a power capable of long-distance transmission, is unquestioned. It is true, it has not yet realized all the hopes and anticipations of its zealous advocates. Serious objections have been raised against it, and many plants have proven failures, yet it has so demonstrated its merits that, with better understanding, it cannot fail to work an entire revolution in the industries.

Electricity may be carried to any moderate distance, in any desired quantity, through a small light conduit on inexpensive supports and with slight loss. It offers great assistance to engineers in utilizing remote cheap sources of power, and is destined to supersede all known methods of power transmission. It is preferable *per se*, and because the difficulties in the actual transference of matter by mechanical means over the intervening distance are great. The difficulties increase with the

distance in any of the systems, but those with electric are less than with mechanical methods, appliances whose great initial cost and low efficiency have hitherto restricted our work. The first cost of a moderate-sized plant is considerable, compared with other modes; but, once installed, it is easily capable of great extension. Its efficiency is high: whereas a fine steam-engine pays out to the recipient belt only 14 per cent of the fuel energy consumed in the boiler, a dynamo will convert fully 90 per cent of the total water-power into electric energy. The former will consume, perhaps, 2.5 lbs. of fuel per hourly horsepower, which is saved to the latter. The first cost may, in certain cases, favor the latter.

The losses from condensation, friction, etc., in the conduction of steam cannot but be great. If the engine operates an air-compressor, the efficiency is reduced from 14 to 10 per cent at least. As a matter of fact, neither steam nor compressed air can be converted into power with a loss of less than 50 per cent of the energy received. Hence from 5 to 7 per cent is the best that can be expected from the use of these expansive fluids, which can never be regarded as serious competitors, except within a very limited scope. Then, compare the cost and inconvenience of large thick pipes required for the conduction of air or steam with the ease and rapidity of laying, supporting, and insulating a mile of wire. Wire rope gives better results; but for distances greater than half a mile it is superseded by electricity, because of the losses by friction; besides, it can transport power to a certain class of appliances only.

On the other hand, electricity subserves practically all the operations of mining: signalling by annunciators or indicators, lighting, blasting, drilling, hoisting, haulage, etc. It neither vitiates the air, as do engines; nor fog and chill it, as compressed air. There is no leakage of power when the motor is not in use, as with other means, and is especially commended when the power is to be intermittently required. A copper wire $\frac{1}{8}$ of an inch in diameter is equivalent to a $3\frac{1}{4}$ -inch air-pipe, or $\frac{5}{16}$ wire rope, for conveying power at average pressures; cost, 1:27:19 for equal lengths.

19. The electrical units are, *Ampere*, *Volt*, and *Ohm*, respectively, measuring the quantity C , pressure E , and the resistance R , of a current. *Ampere* is the unit of current strength, measured by the deposition of metal from a solution (0.017253 grain of silver per second, or 0.005084 grain of copper). The unit of resistance is the *Ohm*, which equals the resistance of a column of mercury 1 square millimeter section and 1060 millimeters long. A *Volt* is the unit of electromotive force (usually written E. M. F.) and expresses the difference of potential, or of electric pressure. Its value is arbitrary, but fixed. One volt will force one ampere through one ohm of resistance. The energy, P , of a current is measured by the product, CE , in Watts (the unit), 746 of which equal a horse-power. The number of horse-power in a conductor equals CE divided by 746.

$$P = CE, \quad E = CR.$$

A Joule, = W , is the work done or the heat generated by 1 Watt in a second.

$$W = QE = 0.7373 \text{ ft.-lbs.}$$

Manufacturers' tables furnish the data for wires of various sizes, by which their resistances may be known. For example, 1000 feet of No. 1 gauge copper wire (0.37 inch in diameter) offer a resistance of 0.1147 ohms and consume 18 volts with a given current of 77.7 amperes.

The electric fluid is conducted by copper wire, the size of which is commensurate with the quantity of energy to be transmitted. The transmission is realized, irrespective of distance, with only a slight loss due to the heating of the conductor and poor insulation. This represents a loss of power which varies with the length and area of the wire. The loss of electric pressure, E , in volts, is equal to the product of the quantity, C , in amperes, of electric fluid to be carried, and the resistance, R , in ohms, of the conduit of given length and diameter. The value of this loss of energy, in Watts, is expressed by $p = CE$. Usually, however, a drop in voltage and the heating limit of the wire are of more consequence than the mere waste of a small fraction of total energy.

The drop in voltage increases inversely as the area or the square of the diameter of the wire, and this is reduced to a

minimum by the use of a large wire, which, however, increases the largest item in the cost of insulation. On the other hand, the employment of a high voltage is possible, the generation of which is only slightly more expensive than that of lower pressure, and such a high voltage can be safely carried by a small conductor when special care is given to its insulation. The economy of electric transmission, as is true also of pneumatic or hydraulic transmission, increases with the pressure employed; for outdoor work, only the difficulties of insulation of a high voltage limit that which may be transmitted on surface lines; for mining work, the question of safety determines the limit of voltage at a maximum of 450, though the fire dangers of a higher tension current may be eliminated without prohibiting the use of a bare wire of ample size. While the allowable expenditure for wire in any given plant will determine the electromotive force which may be used and the efficiency of the plant required will fix the line loss to be allowed, the most economical area of conductor is that for which the annual interest on capital outlay equals the annual cost of energy wasted. In determining the size of wire required for mine work, the allowance for the drop in voltage is about 15 per cent, or even 20 per cent, of that of the current at the generator. The expression in electric units for the required diameter of a conducting wire is

$$d^2 = 21.48DC \div xI,$$

wherein D is the total length of the wire in feet, I , the initial voltage of the current, and x the decimal percentage of loss allowed.

The gain in the use of high-tension electricity is well illustrated by the above equation; but owing to the difficulty, if not impossibility, of preventing leakage at the commutator of a continuous-current generator, the limiting difference of pressure permissible at the terminals of its wires is 1000 volts. With the use of the alternating-current machines, however, there is no limit to the electromotive force which may be given to the current. So that when electric power is to be generated at a great distance from the point of its application,

a high-potential alternating current may be generated and transmitted to the point of distribution at the mine, where a transformer may reduce the high-pressure current to one of a pressure low enough to be safely employed, and, in addition, may convert the alternating to the more convenient continuous current. The employment of transformers has rendered it possible for numerous small mines to install electric plants which formerly were prohibited because of the excessive amount of wire required. At present the cost of erection is far greater than the cost of the wire.

When the current from a locomotive motor is returned by the steel rail to the generator, the rail-lengths are well bonded and their cross-sectional area is at least seven times that of the copper wire which would otherwise be employed, or of the trolley wire.

The alternating-current system, or rather a modification of it known as the three-phase system, offers particular advantages for the transmission of power, and if three wires are used for the current, all varieties of the mine machinery may be driven with the same loss in transmission using the same initial voltage as in the two-wire continuous-current system. In case the continuous current is desired for the locomotives in lieu of the alternating current, a rotary converter can be placed on the line to transform the one into the other, and even also to reduce the voltage to the pitch necessary for the new use.

The conversion of electric into kinetic energy is accomplished by a motor directly connected with fixed or movable appliances (Fig. 32), which may be operated by rotary motion; for the reciprocating motion of pumps and percussion drills, it has signally failed. Rotary drills, fans, hoisters, and coal-cutters are in successful operation, with an efficiency of from 60 to 80 per cent of the energy received. Neither the generator nor the motor is a large or a complicated piece of machinery, being easily transported and run. It therefore admits of introduction within the prescribed limits of the stope or gallery. No power is consumed, and none transmitted to machines which are idle, and the power is always proportional to the work doing. The commercial efficiency of the motor is nearly the

same, whether working at full capacity or not, and it quickly responds to recurrent demands upon it without excessive loss. If the three-wire system is used, then all motors not requiring frequent handling should be connected to outside wires; drills, and the like, to the neutral wire. This plan lessens the pressure on the motors. For lighting, continuous or alternating currents may be used with equal efficiency; but for motors, I

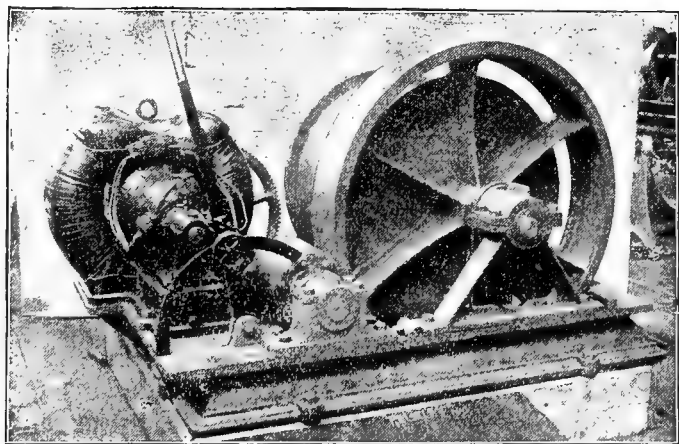


FIG. 32.

do not believe the alternating current can be advantageously used. When one recalls that the current which furnishes the power likewise gives brilliant illumination at the work, one must confess the superiority of the entire discovery.

When the evolution of the storage battery has reached an efficient stage, an important adjunct to mine appliances will have been attained. As yet the storage battery is tentative.

For a proper knowledge of this subject, which is of too extensive a scope for introduction here, the reader is referred to J. P. Jackson's "Electromagnets and the Construction of Dynamos," Kapp's "Electrical Transmission of Power," and Dr. Louis Bell's "Electric Transmission."

20. Water-power has long been employed for operations in the immediate vicinity of the wheel. No cheap and efficient means had been discovered for its transmission to great

distances above the wheel, until the successful enchainment of electricity to man's use. The installation of electricity has opened up the possibilities of water-power to a marvellous degree, with but one disadvantage—the limitations of seasons in drought or cold. The gross power of water is the product of the weight discharged, by the height (h) of its fall. $W = 62.5Q$. The net power is from 40 to 90 per cent of this, according to the kind of wheel used, whether breast, overshot, turbine, or hurdy-gurdy. Q = cubic feet, delivered per minute. Then horse-power = $0.00161Qh$.

In the early days of the undershot and overshot wheels, enormous volumes of water were consumed by large slowly turning wheels, in developing small power. The Leffel and other forms of turbines were next in order. These are quite small, and, revolving at high speed, give a good duty with large volumes of water, under moderate heads up to 300 ft. They may be placed with an axis horizontal or vertical, the largest size being 48", submerged at the bottom of a penstock, or encased in a globe, or cylindrical casing, connected to the bulkhead or piping, by which the water enters centrally and discharges circumferentially. The globe casing with a horizontal axis is the preferable form for mining purposes.

In our mountainous districts the numerous creeks are not large; but their fall, and hence their velocity, is great, and it is rare that water-power can not be found within a moderate distance of the mines. This, the "hurdy-gurdy" wheel has been designed to utilize; and most effectually is it done, giving, as it does, a guaranteed duty of 85 per cent (Fig. 33).

A wheel 18" to 90" diameter, the plane of which may be in any convenient position, carrying a number of small cup-shaped vanes, receives the impingement of one or more jets of water at high velocity, and tangentially. This principle is entirely at variance with the previous methods of generating power, and most nearly conforms to hydraulic laws. Its execution is simple, and a pronounced success, the entire absence of machinery leaves nothing to get out of repair. Placed at the lowest practicable point, to obtain all the head available, the high velocity of even a small volume of water delivered through the nozzles,

will develop an enormous power. Though essentially a high-pressure machine, it is almost as efficient with a moderate

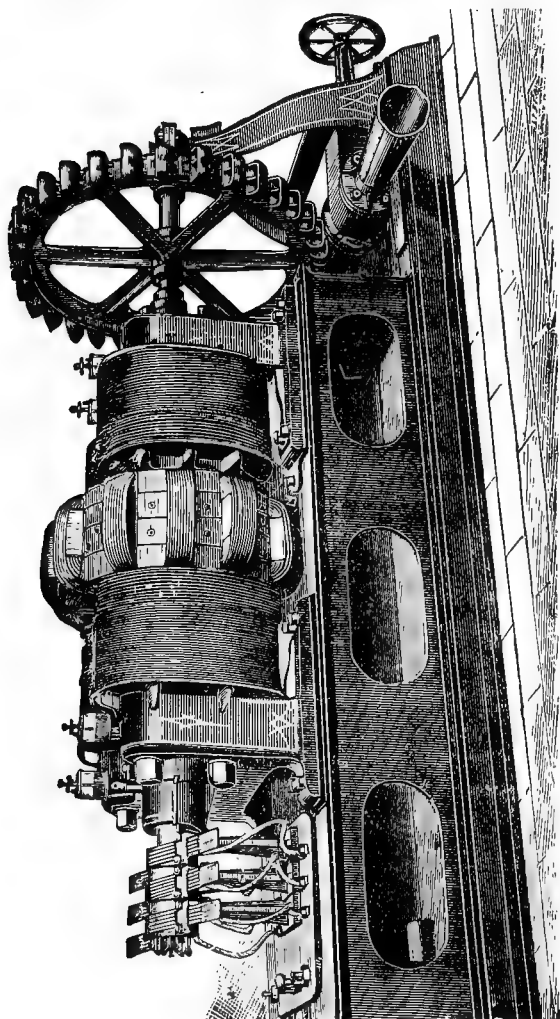


FIG. 33.

head. Thirty feet is regarded as the inferior limit of head, while theoretically there is no maximum limit. It is actually in use with a head of 1,700 feet, and a measured velocity of

revolution of over 7000 feet per minute. The size of the wheel may be proportioned to the rate of revolution desired for the main shaft. A small wheel at high head has a very rapid revolution, and would admit of a direct connection from its pulley to the dynamo, while for slow motion of pumps, air-compressors, etc., a large wheel is desirable.

The Pelton and Knight patterns of wheel are of this type of impulse-motors, and while several European forms are on the market, a hasty glance at them suggests that their parts are not as accessible for repairs as those of the American models, which are durable, reliable, efficient, and easily accommodated to wide variations of power. The secret in the extraordinary energy obtained from the use of these tangential reaction-wheels lies in the fact that the entire dynamic pressure of the water is utilized, as may be shown from the fact that the water falls from the buckets perfectly inert, none being carried over. The manufacturers guarantee an efficiency from these wheels, when properly regulated by governors, to be 85 per cent of the theoretic head due to the velocity of discharge from the nozzle against the wheel cups. The Leffel turbine wheel receives its power from the pressure of a head of water in the ditched penstock, and delivers an efficiency of about 90 per cent, with heads not exceeding 80 feet. It is evident, therefore, that in regions where the fall is great and the volume of flow small, impulse-wheels are to be preferred to the turbine wheels, which, while giving equal efficiency and as much power, require a larger volume at a small head. The water is carried from its source to the wheels either through a continuous line of pipe, which should be as large as admissible, for the first class of wheels, or by a ditch for the second class. The pipes are dipped in tar or asphalt, laid on the ground and strapped or chained to posts or stumps driven for the purpose. The slip-joint connection is better than the ring-joint (Fig. 79). The supply of water is estimated in "miner's inches," which term, while indefinite, represents a flow of about 1.5 cubic feet per minute. By a "miner's

inch'' is known that volume of water which is discharged through one square inch of an aperture which is 2 inches high and 4 inches long, cut through a plank 1.25 inches thick, the lower edge of aperture being 2 inches above the bottom of the measuring book and the upper edge 5 inches below the level of the water.

In order to assist the engineer in determining the size of pipe requisite for a given flow of water, the following formulæ are given. There is no simple relation between the theoretical and the actual discharge of water; for the condition of the inner surface of the pipe, its smoothness and freedom from rivets, laps, and intruding gaskets, and the length of the pipe, are elements affecting the resistance to the passage of the fluid. Elbows or bends in the pipe and variations in its diameter also have an important bearing upon the efflux. The frictional resistances are directly proportional to the length, are inversely as the diameter, and increase with the velocity. The effect of elbows may be separately determined while each change in the diameter, whether of enlargement or contraction, causes a reduction in the flow that may only be ascertained by special investigation.

Assuming, however, very long clean pipes of a uniform size, the resistance due to friction of the fluid in the pipe is expressed by the formula

$$h = 0.1007f \frac{lQ^2}{d^5}.$$

This is usually measured by a column of water, which must afterwards be subtracted from the total head H in order to obtain the effective height that will produce a given velocity or discharge a given quantity. This height or loss of head is h . In clean pipes of smooth bore f is 0.003, nearly, and in the ordinary mine pipes it is taken at 0.0053. The theoretical quantity discharged in cubic feet per second is $6.3d^2 \sqrt{H}$.

But the actual discharge and the loss of head are found to be at variance with these calculated results, and numerous empirical formulæ have been prepared, a substitution in which

gives a closer approximation to the measured quantities. The following are selected (all the units being in feet or seconds).

H is the total head ; h , the loss due to friction in the pipe ; l , the length ; d , the diameter ; Q , the quantity discharged at velocity ; $v = 1.273Q/d^2$. For a certain fixed loss of head, h

$$v = 203 \left(\frac{d}{4} \right)^{0.694} \left(\frac{h}{l} \right)^{0.555} ;$$

$$h = 0.000606l \frac{Q^{1.802}}{d^{4.855}} ;$$

$$d = \left[0.000606l \frac{Q^{1.802}}{h} \right]^{0.206} .$$

Q , for the maximum horse-power,

$$= H^{0.555} d^{2.694} \frac{1}{(0.0017l)^{0.555}} .$$

The maximum horse-power to be obtained is equal to

$$\text{H. P.} = 2.466H^{1.555} \left(\frac{1}{l^{0.555}} \right) d^{2.694} .$$

EXAMPLE 1.—A pipe is 500 feet long and 3 inches diameter. What should be the head to produce a discharge of 180 feet per minute ? 3124 feet.

Here, $Q = 3$, $l = 500$, $d = 0.25$, and assuming f to be 0.00566,

$$h = 0.1007 \times 0.00566 \times 500 \times 9 \times 1024 = 2624 \text{ feet.}$$

EX. 2.—What diameter should it have to deliver the same quantity of water with a head of 82 feet ? 6 inches.

EX. 3.—Required the flow of water through a pipe 2000 feet long, 13 inches in diameter, and with 200 feet head. For the maximum horse-power we have

$$(200)^{0.555} (1.08)^{2.694} \frac{1}{(3.4)^{0.555}} = 0.864 \text{ cubic ft. per second,}$$

and

$$H. P = 2.466 (200)^{1.555} (1.08)^{2.694} \frac{1}{(2000)^{0.555}} = 12.373.$$

Ex. 4.—What would be the loss of head in pumping 2000 gallons of water per minute through an 8-inch pipe 600 feet high? $Q = 4.44$, and $h = 47.6$, or 38.3 feet, according to the equations employed.

Ex. 5.—What horse-power is consumed in overcoming friction in the previous example? Assume h to be 38.3. 19.4 h. p.

The flow is 4.44 cubic feet per second, and the horse-power is $0.1134QH$.

Ex. 6.—What horse-power will be given out by the discharge of 400 cubic feet of water per minute out of a pipe of 13 inches diameter, 2600 feet long, with a head of 400 feet?

The loss of head is 32.53 feet, and the h. p. available is 276.

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CHAPTER VI.

HOISTING OPERATIONS.

21. Hoisting-derricks, construction of; essentials for strength and safety; overwinding, and the devices for preventing the same; indicators, and the modes of communication with the mine. 22. Calculation of the strains in hoisting-frames; constructions in iron and wood; sheaves and their importance. 23. Calculation of the hoisting-capacity of a mine or shaft; hoisting-velocities under different conditions of timbering; loading and unloading conveniences; formulæ and examples; work of the engine in hoisting; definitions of horse-power, indicated, theoretical, and calculated; formulæ; examples.

21. THE most important surface feature is the frame, "head gear," or "derrick," which affords the skilful constructor excellent opportunities to satisfy the two necessary conditions, height and strength; the first for security against overwinding, the second is fundamental.

It is obviously essential that the sheaves on the frame should be placed at considerable height above the ground, to allow sufficient margin within which the engineer may stop the hoist. With the present high speed and large drums, the allowance should not be less than the length of one drum coil of rope, for in a moment's hesitation, or error in the interpretation of the signals, carelessness in signalling, or a derangement of any appliance, the tub or cage may be dashed through the roof before the engine could be stopped. Ordinarily, a brakeman at the mouth of the shaft, having charge of the delivery and receipt of the cars to the mill or dump, may, as the cage approaches the top, signal to the engineer, or the latter may have to depend upon his own watchfulness. So,

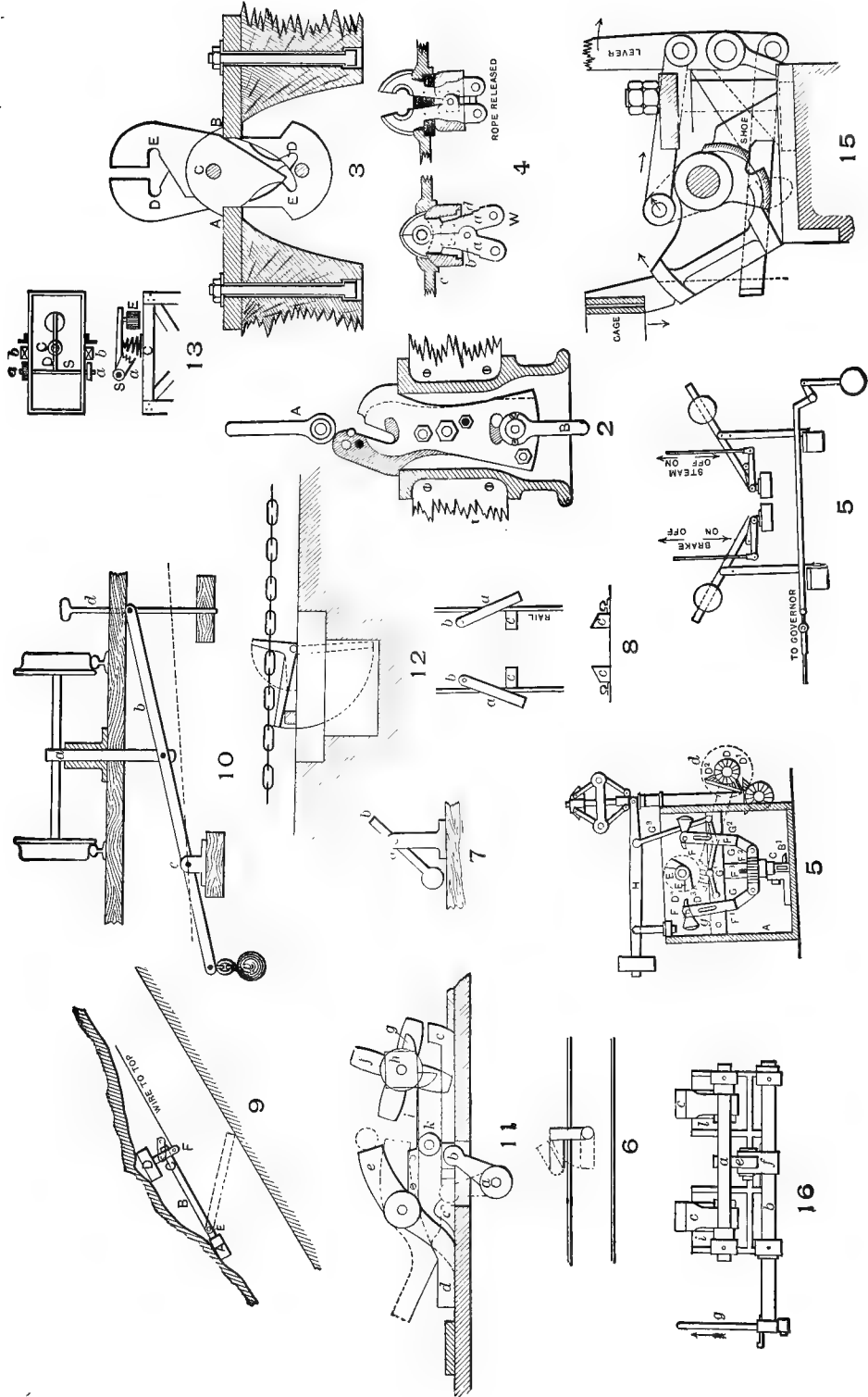


FIG. 40.—SAFETY APPLIANCES FOR SHAFTS AND SLOPES.

devices for preventing overwinding are more or less adopted. But, while desirable, they are not satisfactory. The number of

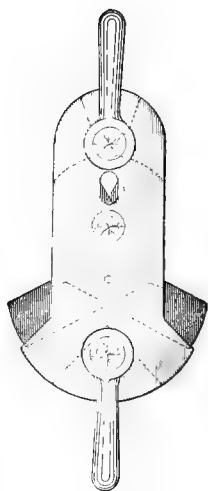


FIG. 34.

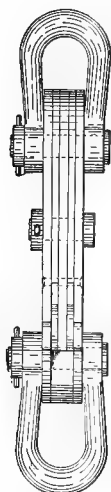


FIG. 35.

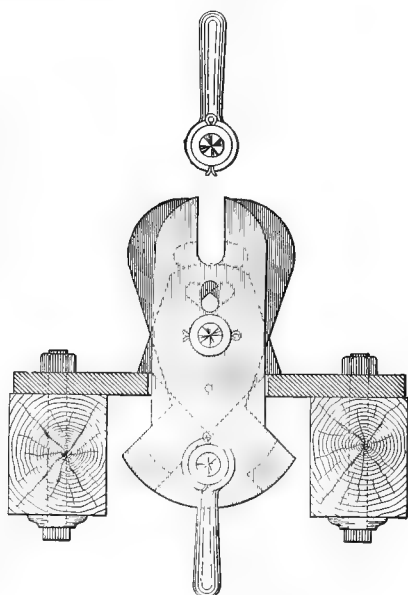


FIG. 36.

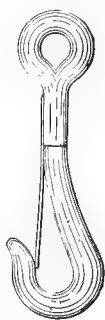


FIG. 37.

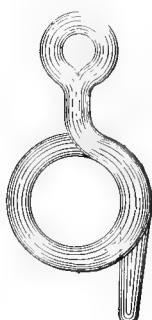


FIG. 38.

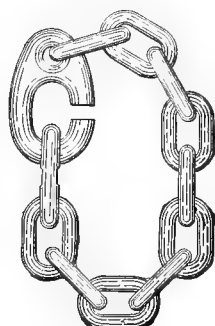


FIG. 39.

casualties are reduced by their use, but they are not wholly prevented. The principle consists in automatically detaching (Figs. 35, 36, and 37) (ice-hooks open and free the cage) or in cutting (a pair of shears cut the rope) the cage from the run-

away rope, when it has reached a dangerous height, and simultaneously throwing out landing-dogs, to catch the falling cage or tub. Or, another plan is to have the guides incline slightly inward. Then the cage, in its ascent, gradually wedges tighter and tighter, and this acts as a brake to the engine. Again, self-acting steam brakes on the engine are constructed so as to operate when the cage reaches a certain point in the shaft. Eternal vigilance is the price of safety, and the only safeguards, after all, are a competent and sober engineer, with machinery in order, a good indicator, and an unobstructed view of the shaft. If the last-named is not possible, a cool, competent brakeman at the platform is indispensable. One device suggested appears to be eminently worthy of introduction,—a 10-ton weight hanging, like Damocles' sword, by a thread, over the engineer in charge, to be dropped when the overwinding has reached the fatal limit.

The same may be said of other forms of safety appliances, even those required by law. They may remedy evils aimed at, but introduce others. First, too great a feeling of security is induced, and negligence results. Second, when the emergency arises, they are rusted or out of order and fail.

The position of the cages in a shaft or slope is ascertained by indicators. An index, operated from the drum shaft by gearing, rope, or worm and screw-wheel, moves around a dial or along a miniature representation of the shaft, at a speed commensurate with that of the cage or tub (Fig. 21). A cylinder has 15 or 20 turns of a spiral thread cut on its face; a pointer moves vertically in the thread as the cylinder revolves, accurately indicating the position of the cage (Fig. 23). The more trustworthy ones are so geared that the index moves faster as the cage approaches a landing stage. A glance suffices to inform the engineer, who need not fear overwinding if brake and throttle are in order. White marks or rags tied on the rope are useless, as also the attempts to make the cage automatically signal its warning to the engine-room.

The safest and most natural means which suggests itself for communication between the engineer and the miner is the

voice, with or without the intervention of the speaking-tube. The telephone or annunciator is more convenient. Then misinterpretations can be excused only by sudden death or criminality. The clumsiest and most unreliable signaling arrangement is the gong-bell or triangle, which is struck by a weighted lever, operated from below by a rope or wire. Its simplicity commends it, while its crudeness condemns it. Mistakes do so readily occur. A stroke of the bell may be lost by too light a pull; or an engineer, anticipating two bells to lower, may not await the completion of the signal, and lower before he has heard the third bell, meaning, perhaps, "hoist with a man on." If a simple uniform code of signals could be agreed upon and adopted by mining men, a great advance would be made. A man formerly accustomed to hoist for one bell, will do considerable damage in his new job, where one bell means to lower. See code on page 545.

On page 108, Fig. 40, are illustrated the various measures for increasing the safety of ascent from and descent into mines.

In No. 4 is Walker's safety attaching-hook, in which a loop encircles the hook and is bound by it to two copper rivets which are sheaved when the hook is down. The jaws then open and release the rope, locking the suspension-jaws on the disengaging-plate.

No. 2 is the Omerod's, and No. 3 Middleton's hook operating on the same principle.

Nos. 5, 5, illustrates the "Visor," an automatic regulator of the hoister whereby the brake is applied and the steam shut off if the engine-speed is excessive at the end of the hoist.

No. 6 represents "blocks" or "stops" of two pieces of wood at right angles to one another moving on an upright pin, one arm being thrown across the track to hold the car in place until it is released.

No. 7 is a balanced block, the end, *b*, of which stops a runaway car on down grade.

No. 8 is the Mortier safety-catch, consisting of an axle with levers attached, placed in the axis of the roadway and

supported upon sleepers. The axle is movable, and in one position is opened by the rising car, but in the other closes after the train passes.

No. 9 shows an arrangement dropped from the roof by a wire to stop the runaway cars on the incline.

No. 10 is a similar appliance with a weighted lever.

No. 11 automatically controls the delivery of cars in and out of the cage, and serves in the intermittent delivery from incline planes or platforms, without the need of attendants to release the cars.

No. 12 is a safeguard placed at the foot of an incline.

No. 13 is a Callow's safety clutch, which is not dependent upon the breaking of the rope to throw it into action.

No. 15 is a falling cage-rest or stop which is operated by a system of levers to release the cage and allow it to be lowered into the mine without a previous lift, as is the case with other cage-keeps.

No. 16 is another form of cage-stop described in Glückauf.

22. The strains to which a derrick is subject are those arising from the weight of the conveyance, its contents, and the rope and friction operating vertically, and the pull of the engine on the inclined rope, which is greater than the weight, by the amount of frictional resistances at the sheave. This latter may be taken as 4 per cent. of the weight. These combine to produce a resultant operating in a direction nearly bisecting the angle between the two ropes, and equal in intensity to about twice the inclined strain, multiplied by the cosine of half the angle between the ropes. A single stick of sufficient size may be placed in this direction, and used as a gin-pole for hoisting; but it is not stable, and, instead, should be a frame, the base of which shall embrace within its parallelogram the line of the resultant. The more nearly central this line falls, the more stable the structure, but the brace then becomes long, and its section large. Still the engineer will prefer an increased stability to a slight saving in material, and hence will allow an angle of about 60° between the ropes.

The form of the frame is essentially two right-angled tri-

angles (Fig. 16), the brace and upright being nearly parallel to the rope. They are set into cast-iron shoes, bolted to the sill; sometimes the posts are tenoned or dovetailed into it. The top frame is slightly narrower than the base, which consists of a sill on each side, connected by three cross-sills mortised and dovetailed to them, the whole bolted and anchored to heavy "dead men" buried in the ground. Their risk of fire, the exposure to weather, the working of the joints, and the difficulty of securing sound, long, large sticks, render the adoption of other material than wood highly necessary. Wrought-iron is much used. Height is the essential feature of derricks, but this, with stability, is difficult of attainment without a rigid frame, perfectly made. This can better be secured by the use of Phoenix or Kellogg columns, set in cast-iron shoes bolted to heavy masonry pillars, Fig. 42, or by well-tied timber, Fig. 41.

Greater stability can be secured for the sheave, by building, of wood or iron, a framework of four vertical posts at the four corners of the hoisting compartments, suitably braced and tied with struts from the top, inclined slightly less than the resultant force, the lower ends being stepped into masonry pillars, or joined to a substantial base frame. Fig. 36 is an example.

Upon the top of the uprights is mortised a frame supporting the sheave with its axle horizontal, and its unsupported length as short as possible. The diameter of the sheave should be 100 times that of the wire rope (48 at least), owing to the rigidity of the rope, which resists bending. To minimize this resistance, the wires of the rope are as fine, and the angle of the bend as small, as obtainable.

This is more imperative as the speed of hoist increases, and not uncommonly sheaves are seen 12 feet in diameter. All pulleys over which the rope bends more than 30° become, to all intents, sheaves. The hubs are double, connected to the cast-iron rim by wrought-iron rods let into sockets, and they should be as light as practicable at the rim, because, by reason of the impetus they acquire, they continue to run long after the hoisting has ceased. Often the sheaves are cast in sectors, afterwards bolted together. The grooves are lined with wood,

on end if possible, and tarred hemp to prevent slip; for the transmission of power, with rubber.

Never house the derrick, especially about collieries, for, in the event of a fire it becomes a draught-chimney (Fig. 42).

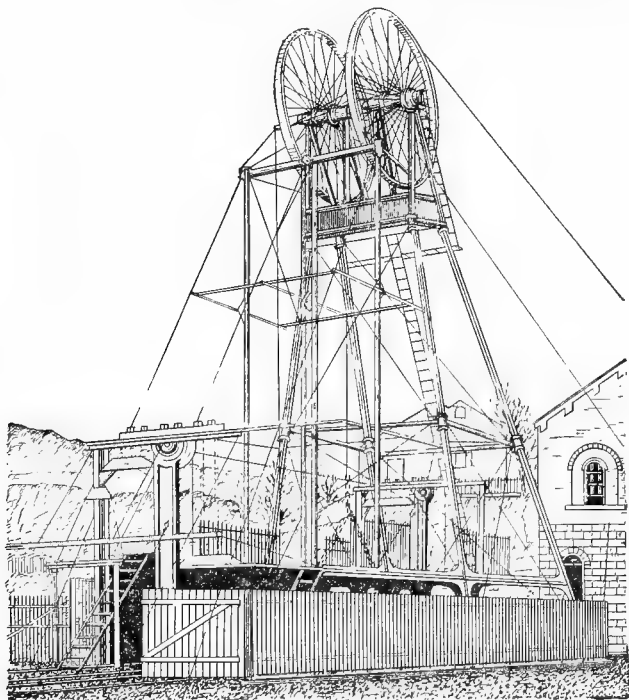


FIG. 42.

23. As the mining engineer may find himself compelled to calculate the plant required, we will briefly consider the conditions and the process. The output of the mine depends upon the time occupied in each hoisting-trip and the load carried, assuming that the conveniences for delivering to the shaft at the bottom, and the facilities for the disposal of the ore and its carrier, are equal to, if not greater than, the hoisting capacity. The speed of hoist is limited by the equipments of the shaft, which must be timbered very substantially to permit

rapid hoisting. Cages are being hoisted in vertical shafts, at rates up to 2500 feet per minute; skips and slope-carriages at 1,000 feet; and buckets at not over 300 feet per minute. The time allowed per trip must also include the arrangements for loading and unloading. The time lost in filling an attached bucket at the bottom, and dumping it at the top, is from three to five minutes; if the empty buckets are immediately replaced by full ones, much less at both ends. A car can be run on and taken off a cage or slope carriage in twenty-five seconds. A skip occupies from two to three minutes to side-track, unload, and return. The influence of this loss of time can readily be calculated.

Let t = the minutes to load and unload ;

D = the depth of the shaft in feet ;

v = velocity of the hoist per minute ;

v' = " " lowering " " ;

n = number of trips per hour ;

T = minutes per round trip ;

Q = output tons per hour ;

q = load tons per trip ;

$$T = \frac{D}{v} + \frac{D}{v'} + 2t ;$$

$$n = \frac{60 v v'}{D v' + D v + 2 t v v'} ;$$

$$q n = Q.$$

Various transformations may now be made according to the known conditions. Usually q is given, and it is desired to ascertain Q .

Thus, in a poorly timbered shaft, if only one bucket be run up and down without detaching, the output from a 300-feet shaft is about 3.6 tons per hour. With 3 buckets in constant use, each holding 600 pounds, the hourly product cannot exceed 6 tons. With excellent timbering, double the speed may be permitted, in which event 3 buckets will deliver 7.2 tons per hour at the surface.

So it is evident what a large proportion of the time may be lost at the landings, when even the doubling of the speed only increases the quantity one fifth. Skips loaded from shutes are almost as wasteful of time. Hence, for large mines, cage and cars are resorted to. Then, from the 3000-feet level, 36

tons may be hoisted per hour, by three cars in constant use, holding each 3,000 pounds, and assuming a not uncommon rate of 1,800 feet per minute.

The size of the engine must necessarily depend upon the velocity of the hoist, the load, the dead-weight of the rope, cage, car, etc., and the various resistances. Unless there is a counterpoise (see p. 90), the maximum work of the engine is at starting, when the inertia of the load, M , is to be overcome.

R = the weight of rope per foot ;

B = weight of bucket, car, cage, etc.

Then $M = 2000q + RD + B$.

The resistance of friction, etc., is about 12 per cent with cage ; 4 per cent with buckets ; and 20 per cent with skip. Therefore the value of M is greater than that given by 4, 12, or 20 per cent, as seen ; and equals

1.04, 1.12 or 1.20 times $(B + RD + 2,000q)$.

Though the load, M , is not operating throughout the hoist, it is necessary to have sufficient power to start as quickly as possible, without jar. Moreover, a force nearly twice M is required to overcome the inertia of the load. So the allowance is made as indicated, though it is indeed too small for the initial stages of the hoist, and too great during the final.

The work done is always a product of the resistance M , (the strain on the rope) and the velocity per minute, the horse-power, H , being found by dividing by 33,000.

Thus we require theoretically 754 horse power to hourly raise 36 tons from the 3,000-feet level by a cage and car weighing 2,300 pounds and rope. With a tapering rope, 150 horse-power may be saved. With double cylinder drum, two cages and cars, the same product may be raised at 1,300 feet per minute, and only 430 horse-power needed.

The term horse-power of an engine has a three-fold interpretation :

1. The *indicated* horse-power, wherein the actual work done is measured by an indicator, the friction by a dynamometer.

2. The *theoretical*, which is the product of the boiler pressure, the area of the cylinders and the piston speed per minute ;

the losses from imperfect delivery of steam, friction of valves, etc., clearance or compression are not considered.

3. The *calculated* horse-power is a certain fractional part of the theoretical, found by multiplying the latter by a modulus, m , which allows for the losses mentioned as well as for the intermittence in running. The value for m varies from 0.60 in a very poor engine to 0.90 in one in good working order.

If s = the length of piston-stroke in inches; k = the diameter of cylinders in inches; N = the number of revolutions per minute; $0.166Ns$ = the piston-speed, feet per minute; c = the number of cylinders; f = the coefficient of friction; p = mean effective steam (gauge) pressure, pounds per square inch; P = steam-pressure (gauge) on entering the cylinders; H = the horse-power actually being performed by the engine,—then

$$H = \frac{c p s N k^2}{252100} m, \quad 33000H = (2000q + RD + B)(1 + f) \frac{Dn}{30 - tn}.$$

From this latter equation it is possible to calculate the depth to which a given engine will do service, for all the quantities are known, or may be assumed, except D , which is then calculated. See examples.

With a plain slide-valve engine using steam throughout the entire stroke, the boiler-pressure may be substituted for p without sensible error. If, however, the indicated horse-power is measured, both H and p are very accurately known. It must be remembered that the above gives only the approximate work of an engine, without considering the influence of the masses of the rotating and reciprocating parts. Generally speaking, the heavy revolving pieces tend to equalize the speed of the engine and the load; but the varying angularity of a short connecting-rod influences the rotation of the crank-shaft in such manner that it is faster during the first part of the outward stroke and the last part of the return than during the remainder of the revolution. In like proportion, the crank-pin receives a varying rotational effort which is *nil* when the piston is at either end of its stroke, and a maximum at some certain point about 80° to 100° from this.

For engines in constant duty, the inequalities may be cor-

rected by a sufficiently heavy fly-wheel without any material loss of work. In lieu of a fly-wheel the hoister has the inertia of the drum; and the greater the drum, or the length of the connecting-rod, the more uniform is the velocity.

Hoisters, moreover, as stated on p. 66, must be capable of *starting from any point of the piston-stroke*. To be assured of *starting* the load as well as keeping up the velocity, the hoister is built with two cylinders having their cranks set "at quarters," as mentioned on page 74. To determine the conditions of equilibrium by moments, it must be remembered that the pendent weight M , with its lever-arm equal to the half-diameter of the drum, r , is to be in static equilibrium with the sum or the rotational forces on the two cranks multiplied by the crank-arm (one half the piston-stroke).

To facilitate calculations the following table has been computed. It gives a coefficient, C , representing the maximum and the minimum moments for duplex engines with various expansions and ratios of connecting-rod, l , to crank-arm, a . The calculations are made on an assumed clearance of 7 per cent. Then $CsPk^2m = (M + Mf)r$.

Apparent Cut-off.	$l = 4.5a.$		$l = 5.5a.$		$l = 7a.$	
0.00	0.5615	0.3974	0.5600	0.3974	0.5580	0.3974
	0.4664	0.3633	0.4606	0.3688	0.4586	0.3696
	0.4380	0.2472	0.4365	0.2499	0.4353	0.2550
	0.4256	0.1878	0.4193	0.1923	0.4124	0.1966
	0.3893	0.1469	0.3818	0.1473	0.3786	0.1512

Cog-gearred, second-motion hoisters may be calculated similarly by introducing the proper coefficient, C , in the following formula, where x and y are respectively the number of teeth in the pinion and drum-wheels:

$$CsPk^2ym = M(1 + f)rx.$$

The data given here may serve as a guide for shaping the engineer's opinion as to the hoisting capacity required. The following carefully compiled tables may assist in illustrating the benefits derived from the expansive use of steam. The first column is the ratio of expansion (the ratio of the volume finally occupied by the steam to that at the moment of cut-off).

assuming a piston clearance of 7 per cent of the cylinder capacity; the second gives the corresponding period of cut-off; next is given the average mean total pressure (absolute) on the assumption of perfect expansion in accordance with Boyle's law; the fourth contains the total actual work done by one pound of steam at 100 lbs. initial pressure; finally is given the consumption of steam in pounds per horse-power of actual work done per hour.

Ratio.	Cut-off.	p	Work per Pound of Steam.	Steam per Horse-power.
6.30	$\frac{1}{10}$	0.413	151,370 ft.-lbs.	13.08 lbs.
5.50	$\frac{1}{8}$	0.457	146,323 "	13.53 "
4.50	$\frac{1}{6}$	0.527	138,130 "	14.34 "
4.00	$\frac{1}{5}$	0.567	132,770 "	14.90 "
3.35	$\frac{1}{4}$	0.637	124,066 "	16.00 "
2.65	$\frac{1}{3}$	0.726	112,220 "	17.70 "
1.88	$\frac{1}{2}$	0.860	94,200 "	21.00 "
1.00	1	1.000	58,273 "	34.00 "

No allowances are here made for the results of wire-drawing, pre-release, or back-pressure. If the steam is not at a boiler-pressure of 100 lbs. per square inch, the following list of multipliers will serve for calculating the actual hourly performance of work under other initial pressures with their corresponding temperatures:

P	Temperature of Steam.	Multiplier.	P	Temperature of Steam.	Multiplier
70 lbs.	302°.9 F.	.981	110 lbs.	334°.6 F.	1.000
80 "	312.0	.988	120 "	341.1	1.011
85 "	316.1	.991	130 "	347.2	1.015
90 "	320.2	.995	140 "	352.9	1.022
95 "	324.1	.998	150 "	358.4	1.025
100 "	327.9	1.000	160 "	363.4	1.031

Ex. 7.—If $s = 48$, $N = 40$, $p = 60$, and $c = 2$, the engine required for the example on p. 117 is one having a cylinder of 28×51 inches, allowing for 7 per cent clearance.

Ex. 8.—Required the size of an engine to hoist 1000 tons per 10-hour shift from a shaft 1200 feet deep. The cage load being 4 tons, $P = 100$ with a cut-off of $\frac{1}{4}$, $N = 40$, and a modulus of 0.80.

$$Q = 100, \quad t = 0.4, \quad D = 1200, \quad q = 4.$$

Let $v = 1500$, then $T = 24$ and $n = 25$ trips. Whence

$$M = 8000 + 1500 + 800 + \text{rope.}$$

if the rope weighs 2 lbs. per foot, then $0.01615k^2 = 646.5$, or k is nearly 27 for $s = 60$ inches.

As 1200 feet of rope must be coiled on the drum at a rate of 1500 feet per minute, the direct-connection drum making 40 revolutions must have a diameter of 12 feet.

Ex. 9.—What load will this engine start?

For a length of connecting-rod of 165 inches (or 5.57) the minimum coefficient is (p. 119) 0.1923, whence

$$0.1923 \times 5 \times 100 \times 729 = 1.12M \times 6,$$

and $M = 10,386$ lbs. In order to start the load of 12,700 lbs. calculated above, the boiler-pressure, P , should be 122 lbs., or else the engine must be stopped at a point different from that corresponding to the minimum effort on the crank-pin. A drum of 59 inches radius would give equilibrium, but then the velocity of hoist would be reduced to 1236 feet per minute during the early period of the hoist, though it would gradually increase to nearly 1500 feet per minute.

Ex. 10.—If it be required to know to what depth a given engine will do service, the order of procedure is as follows: Substitute in the second equation on p. 118 all the known quantities, and solve with respect to D .

Thus, a single 16×30 inch engine at 72.6 lbs. mean effective pressure, and a piston-speed of 300 feet, will give 88 h. p. if its modulus be 0.80. Assuming a cage at 1500 lbs., car at 750 lbs., and a load at 2300 lbs., $M = 4550 + 1.58D$.

If a rope of 1.58 lbs. per running foot be taken, we have

$$88 \times 33000 = (4550 + 1.58D)(1.12) \frac{Dn}{30 - tn}.$$

If the output is 60 long tons per shift, q being 2300 lbs., then, with $t = 0.4$, n becomes nearly 6, because $Q = 6$ long tons (13,440 lbs.). Substituting in the above, D becomes equal to 1704 feet. If $t = 2$ minutes, $D = 1200$ feet.

Ex. 11.—For an output of 300 short tons daily the same engine will do service to depths of 1240 and 829 feet, respectively, under the conditions named below:

$$Q = 15; \quad t = 0.5; \quad n = 8; \quad \text{and} \quad p = 102 \text{ lbs.}$$

$$Q = 15; \quad t = 1.0; \quad n = 5; \quad \text{and} \quad p = 72.6 \text{ lbs.}$$

In each case q is 2 short tons and $R = 2$ lbs., the car and cage being assumed at 3000 lbs.

Ex. 12.—It is intended to erect a pair of coupled engines directly connected, to hoist two cars carrying 3100 lbs. each, a cage weighing 1500 lbs., in a shaft 1800 feet deep, at an average speed of 1000 feet per minute. If the engines are to be $\frac{1}{3}$ cut-off, required their size with a flat rope.

Assume the weight of each car at 1000 lbs. Then the rope supports 9700 lbs. besides its own weight, which is 5 lbs. per foot at the top and for 600 feet down, then $4\frac{1}{2}$ lbs. for each of 400 feet, and 4 lbs. per running foot for the remainder. Each cylinder must be capable of exerting 402 h. p. if the modulus of the engine is $\frac{2}{3}$. Let the boiler-pressure be 80 lbs. effective, then the average pressure is 58 lbs. (p. 120). Each cylinder will be 38 inches diameter and its piston have an average speed of 200 feet per minute. Suppose the stroke to be 4 feet, then 25 revolutions are made per minute by the crank-pin, and also by the reel. The smaller diameter of the reel is 30 inches, and the final diameter, with the last coil of rope, is 147 inches. The sizes of the rope are $4\frac{3}{8} \times \frac{1}{8}$ inches, $4\frac{1}{2} \times \frac{1}{8}$, and $4 \times \frac{1}{8}$. If two reels are employed the cages will pass at a point 1118 feet from the surface.

An engine set to $\frac{1}{3}$ cut-off is working under 90 lbs. boiler-pressure. Theoretically, it should give $122,220 \times 0.995 = 106,609$ ft.-lbs. of work for every pound of steam used, and should consume only 17.7 lbs. of steam per hourly horse-power. If, however, the gear be set to $\frac{1}{3}$ cut-off, 131,224 ft.-lbs. of work are obtained from the same amount of water evaporated and the same fuel consumption. This would correspond to a gain of 23 per cent in power.

It is recommended to engineers that they watch and frequently examine the performance of their engine by the use of the indicator. This appliance is, figuratively speaking, the stethoscope to an engine, and when the merits of it are once tested an engineer will acknowledge it invaluable. It is not sufficient that the engine does not "knock" or that no leaks are apparent, for there are many serious causes of loss, such as too tight stuffing-box, loose piston-packing, bad setting of valves, restrictions in the steam-passages, wet steam, and improper amount of steam compression, which may be unsuspected, and nothing but an indicator will reveal these faults, which may then be easily corrected. The writer cannot impress the value of this little machine too strongly. It is cheap (\$30), simple, readily attached, and requires no elaborate calculations. The results are pictorial. There are a number of works on the indicator which fully explain its manipulation.

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CHAPTER VII.

HOISTING-CONVEYANCES.

24. Kibbles and buckets, their sizes, etc.; objections to buckets in hoisting; guides, etc., for rapid hoisting; skips and gunboats for slopes; automatic dumps and brakes. 25. Slope-carriages compared with skips; cages for vertical and inclined shafts; single- and double-deckers; safety appliances and clutches discussed; landing-doors, dogs, etc., for cages; ropes of hemp, iron and steel wire, round and flat; locked wire ropes; tapering ropes for equalizing the work of the engine. 27. The life of a rope, its care and preservation; splicing and testing; cost of ropes.

24. THE ore is conveyed from the workings to the shaft in kibbles, buckets, or tubs, on small platform cars, or in small box cars. Tubs are of stout barrels, heavily coopered, and supplied with a bale of $\frac{3}{4}$ -inch round iron, hooked into eyes on a strap which is bolted on both sides and under the bottom. A liberal supply of these, equal to the capacity of the hoister, is required for constant use and for emergency. A snap-hook, fastened on the hoisting-rope, catches into a ring of the bale for hoisting (Fig. 32). If, however, the tubs are frequently detached from the rope, serpentine hooks are used instead (Fig. 33). A length of $\frac{3}{4}$ -inch round Norway iron is turned, at one end, into a small ring (in which the rope is to be fastened), the rod is softened and bent to make one spiral turn of 3" diameter, leaving the end free and open.

Buckets are used where the developments are not of an extent to warrant more elaborate arrangement, and also during the sinking of the shaft. Two features are observed in the designing of kibbles: lightness, to keep the size and weight within limit of easy handling by one man, and to reduce the dead-weight in hoisting; and capacity. They are plain or bellied

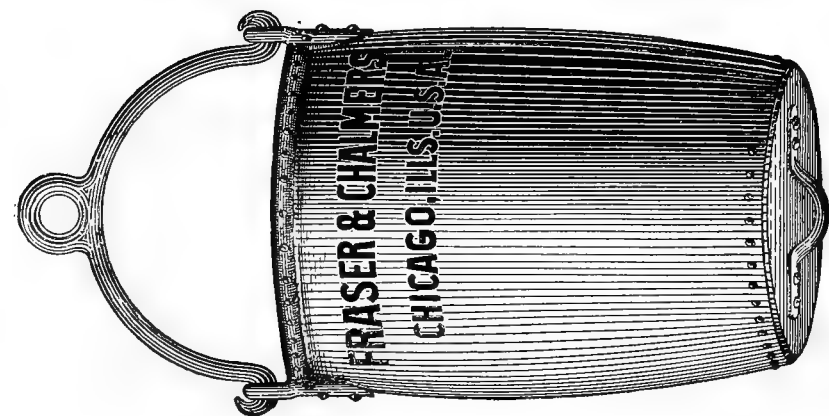


FIG. 44.

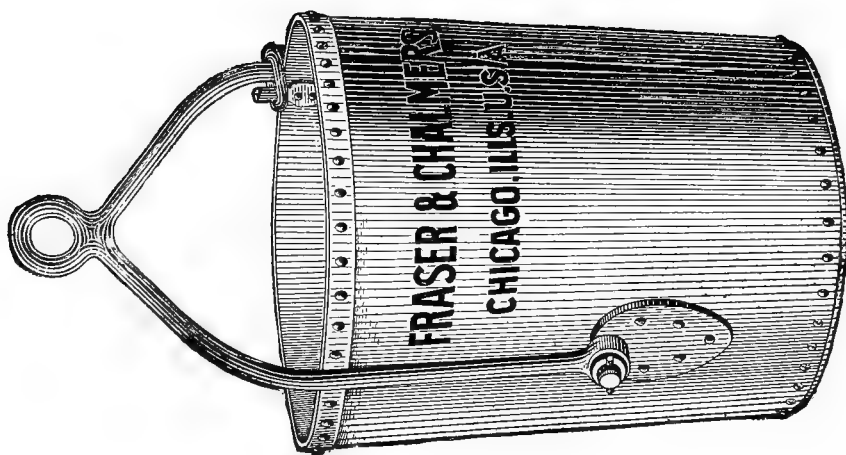


FIG. 43.

cylinders of boiler iron, 18" to 33" diameter, 30" to 54" deep, and weigh from 150 to 900 pounds. They carry from 600 to 3,000 pounds of mineral, and are hung on the rope by an easily-disengaged hook. If wire rope is employed, a short length of heavy chain intervenes between the hook and the rope socket; without this flexible attachment, the wires break at the socket in a very short while. Where a large number of buckets are in use, the empty bucket is replaced by a loaded one, which, on reaching the surface, is immediately unhooked and dumped while another is being lowered. The expense for such an outfit would be high for a mine having several stopping gangs, in which case a very large bucket is permanently attached to the rope, being filled from cars or a chute below, and emptied above by various means. The bucket (Fig. 44) may have a becket and eye underneath, by which the surface man swings it to and fro till he can catch into the eye a short length of stationary chain. The tub, with its hoisting-rope, is lowered until it is upside down; then it spills its contents into a car or over a grizzly. Another device (Fig. 43), not so safe, consists in having the bale pivoted a little below the centre of gravity of the bucket, which is held in position by a loose ring on the bale, slipping over a stout pin at the upper rim of the bucket. To dump, the brakesman merely slides the ring up a little, and the bucket turns over automatically. It is easily righted and fastened again.

Hoisting by buckets is slow and insecure, besides producing and losing much fine material with each handling. This is a loss in coal and even in ore, for, if the fine stuff reaches the surface, it never arrives at the smelter; much value is therefore lost, for the silver minerals are usually the softer. The speed of the hoist can never be large, because of the liability of collisions, unless each tub compartment of the shaft is smooth-lined. Even then the lining must be watched. The end of a plank which has become loose will project and often trip the tub, resulting in great damage and frightful accidents. In an incline the tub slides between a pair of "skids,"—planks laid on the floor.

In the endeavor to employ buckets for quicker hoisting, and to prevent collisions, rails or wire ropes are laid in the shaft. The bucket was suspended from a yoke that slides up and down the guides. Iron rails were abandoned because too "clattery." Wire rope kept taut by screws is feasible, but it has no advantage over wooden guides; besides, as has been noted in previous lecture, the better line of improvement is in the conveniences for loading and unloading. No simple safety appliances are applicable to tubs. The wooden guides are of 4×6 scantling, spiked end to end on the shaft timbers. For cages they must be narrower, and trimmed at the landing-stages so that the safety clutches do not take hold.

For hoisting in inclined shafts, skips, or gunboats, are commonly seen. A strong iron box weighing 900 to 1500 pounds is set on four wheels, held by bosses riveted to the sides. Its cross-section is rectangular, but its side view is a trapezoid (Fig. 45). The inclined end is uppermost, with automatic dumpers, while in those discharging by door it is below. The hoist-rope is attached to the bale which rotates on a pin passing through the side of the skip, often back of the centre of gravity, so as to dump automatically. The charge of one or two car-loads is shovelled or shuted into the skip, which empties at the surface into a bin or on grizzly. In one variety the contents are discharged by the mouth at which it is loaded, while the other form has a swinging door opening at the upper side. A vertical safety skip is shown in Figs 46 and 47.

The automatic dump is simple, the rear or lower wheels are of wider face than the fore wheels. As the dumping plat is reached, the guide-rails on which the wheels have been travelling, gradually bend to horizontality, and these the front wheels follow. As the hoisting continues, the wider rear wheels catch and roll on a pair of outer guides, and continue up the slope. By this means the lower end is elevated and the skip emptied.

The brake is generally a drag, consisting of a bar about 4 feet long, trailing on the floor, and only catching if the skip breaks away on its up trip. Often the wheels are confined

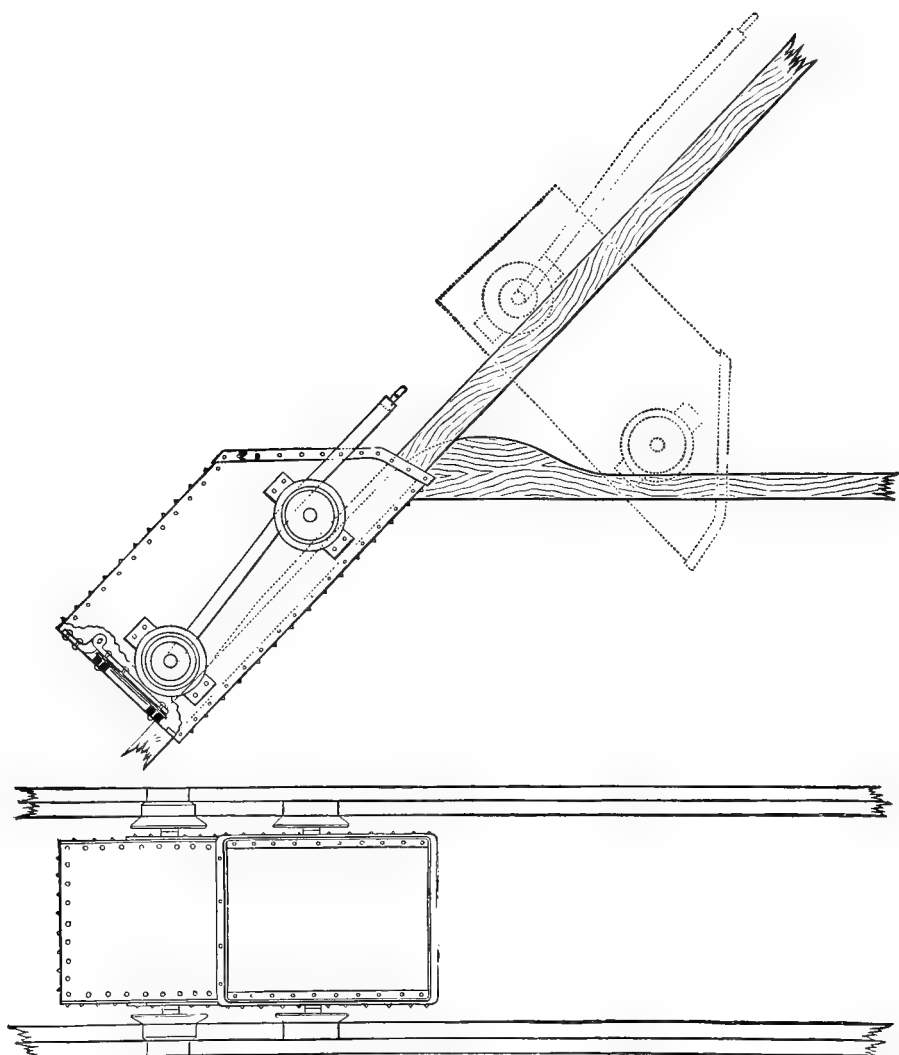


FIG. 45.

between two guides on each side as, for instance, where the direction of the slope changes. The sole objection to these skips is the double and treble handling involved. The car

from the mill-hole empties into the chute, whence the skip is loaded, and at the surface the reverse operation takes place.

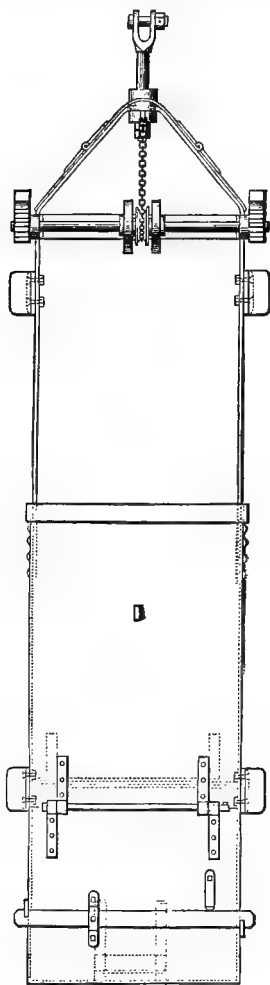


FIG. 46.

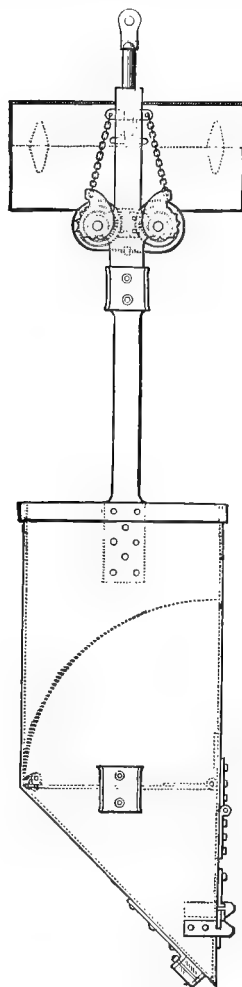


FIG. 47.

25. A slope-carriage dispenses with two handlings, taking the car at once to the surface. This is simply a double triangular frame, large enough to accommodate a car with

two rails on its horizontal top, and two wheels on each hypotenuse. A hook or lock holds the car while riding. For convenience, the loading and unloading gangways are not on the same level, the track for "empties" being 6 feet higher than

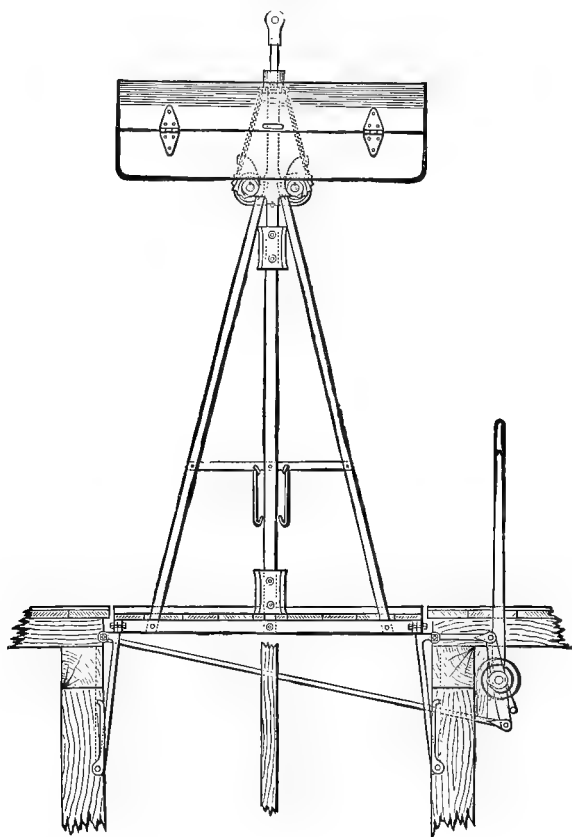


FIG. 48.

the "full" track. If the seam is too thin to allow of this, each track has a curved roadway connection with the gangway.

The head-room required for the slope, having this means of haulage, implies expensive timbering and large area, particularly as double trackway is necessary for a considerable

output. There is not much necessity for a carriage, except in slopes over 30° .

The cage is now almost exclusively used for raising large output from great depths. It is a simple elevator platform, accommodating one or more cars, held on the sides next to the guides by two stout iron frames, united at the top by a cross-bar, to which the hoist-rope is attached (Figs. 42 and 43). Two iron ears at the top and bottom on each side confine the cage to the guides, and, generally, an iron roof or "bonnet" over the cross-bar shields the passengers from falling rocks. The size of the car governs that of the cage, which just fits the compartment. A latch-lock holds the car in place on its journey. Cages are preferably of iron, because the required strength is obtained with less increased dead-weight, though they are not so easy to repair as wood, and accidents arising from "jamming" in the shaft are frequent.

This is the safer, quicker means of hoisting, consumes less time in loading and unloading, and involves less handling of mineral, but demands a well-timbered shaft, otherwise the swelling of walls or bulging of timbers interrupt a steady hoist. A small plain cage

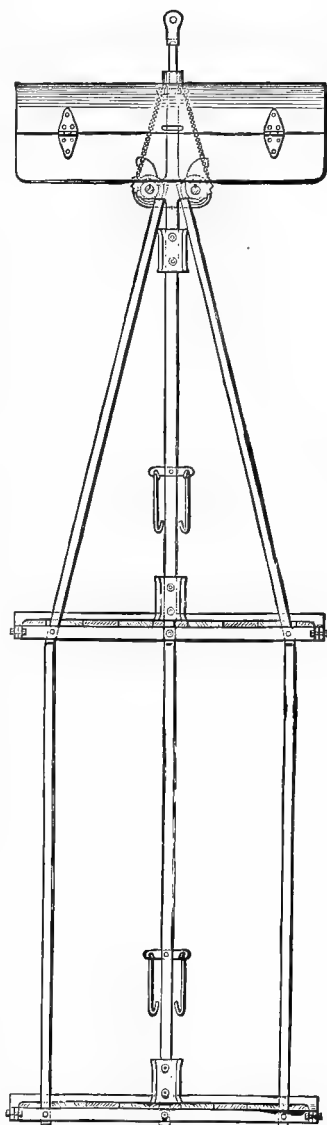


FIG. 49.

costs about \$150; the Nevada safety pattern, \$400. The self-dumpers are cumbersome, complicated, and not a success.

Cages are built with one, two (Fig. 49), or three stories. Single-deckers are almost exclusively used in America, and are sufficient except for narrow, deep shafts, when heavier loads are necessary for large output. The landing-stages are then arranged in the same number of tiers, from and upon which cars are simultaneously run, without moving the cage. With ample facilities for "decking" the cars, the saving in the trip-time, per car, improves the capacity of the plant. An objection prevails against multiple-deck cages, in the necessarily complicated underground stations. The single-platform two-car cage is relatively heavier than the single- or double-decker.

For inclines of uniform slope, as well as for vertical shafts, cages may be had, the platform being hung on an adjustable lever; but the carriage is better.

About fifty years back, various contrivances began to be proposed for guarding against accidents resulting from the breaking of the rope. The sudden starting and stopping of the hoist shock and strain the cable more severely than does the mere weight. This inevitably shortens its life. Without careful hoisting and a heavily timbered shaft the rope cannot be insured against the jars which soon rupture it. To avoid the accident sure to follow, some form of safety appliance must be thrown into action, and arrest the fall of the cage or skip. The safety catches differ in design and efficiency, but depend generally upon a spring, so held between the rope and cage as to be compressed, while the weight of the cage strains the rope, but acts on a clutch that grasps the guides and stops the fall, if relaxed by rupture of the rope or otherwise. The clutch is either a pair of sharp-pointed steel levers, which are thrust outward into the timbers, or a serrated cam, the wider part of which will be turned against the guides, and clutch them on either side (Figs. 46 to 49). The heavier the weight the better the bite, after they once take hold. Many a life has been saved by them, but in many instances also they have failed to operate. A momentary check, or any sudden change

in the speed often unnecessarily throws them into action. On the other hand, they are rarely, if ever, in order when the emergency arises, or the guides are so wet and dirty that the clutches fail to catch, if the momentum of the falling cage is great ; besides, they are costly and troublesome. Though useful adjuncts, which the law requires, yet it is not surprising that the distrust of them is strong ; they breed other and more serious causes of alarm. The tendency following their use is a lack of attention to the condition of the rope and an undue extension of its working life. Fortunately, the rope more often breaks at the moment of starting at the bottom than at any other time, and the point of rupture is where the rope enters the socket. If, however, the rope snaps as it turns the sheave, and this is of common occurrence, there is nothing to prevent the inevitable and frightful calamity that follows—the entire length of the rope falls and crushes cage and contents. A simple appliance might be introduced at the top. After all, the employment of the best materials and men, the careful supervision and repairing of the plant, are the only safeguards.

At the several levels in metalliferous shafts the landing of the buckets is effected on a hinged door, of double 2" planks, lined on top with iron. This is swung against the far side of the compartment, closes it completely, and standing as it does at 45°, the bucket slides into the drift. When not in use, it is hooked upright, closing the mouth of the drift, and leaving the hoistway clear. That miners may go from one drift to the opposite, an escapement roadway is provided around the shaft in the rock. At the mouth of the shaft are two heavy doors, opened for or by the rising bucket, and closed at all other times. These are convenient, and prevent accidents arising from stones dropping down the shaft, especially while dumping the bucket. A similar arrangement in slopes—a movable drawbridge, lowered at pleasure—receives the car for attachment to, or after detachment from, the rope, and closes the gangway when the car is above the plat.

At the surface the cage is held by dogs or chairs (Fig. 48).

These are merely steel levers, projecting slightly over the shaft, which are lifted out of place as the cage rises, but immediately return to position to support it while in use. When placed at the various stations down the shaft, they are manipulated by lever, and thrown into the shaft only as desired. They are simple and convenient, if not indispensable..

26. Ropes used for hoisting are of hemp, aloë fibre, iron or steel wire. The essentials are flexibility and strength, combined with a small weight. Hempen ropes have until recently been entirely used for hoisting, but great caution is necessary in the selection. The practice is, and has been, to work up into new rope fibres and strands of worn-out ropes. This is a serious mistake, for long service cannot be expected of short fibre. Another objection is the increase of weight, while working, from the absorption of moisture. Tallow is its only preservative. Again, the size of the rope for big work is so large, that several round ropes are sewed together into an unwieldy mass, to obtain a sufficiently strong cable. The coiling and uncoiling of one so thick entails great wear, and is expensive.

So wire rope was substituted. It is as pliable as hemp, is much stronger for the same weight and same money, and now has survived the early objections to its introduction, i.e., the great damage predicted from its use in shafts not properly timbered, and that it would not give as early warning of its breakage as does hemp.

The usual number of wires to the strand is 19, twisted about a centre of hemp, which gives a better wearing, a more flexible rope, and one less liable to snap than those of 7 wires, which are used only for guy-ropes, etc. Several twisted strands compose a rope. The twisting gives elasticity to it, and a short twist is advisable for those liable to shock. If the twisting has been conscientiously done, the individual wires of the rope will be equally strained while in use. Its tensile strength, equal to a sum of the strengths of the individual wires, is greater than that of the rod of equal cross-section and

material, the extra manipulation of drawing the wires having increased their strength.

Locked wire ropes are giving good results a great flexibility is imparted to them without in the least impairing their ability to withstand friction.

Steel has a great advantage over iron by reason of its lightness and high elasticity ; besides, it has a slower wear, but it corrodes faster than iron in wet shafts, and if tempered highly breaks quickly. Plough-steel, the highest grade of steel made, is the strongest ; but soft steel is more flexible.

The constant winding and unwinding of the rope over sheave and on drum diminishes its strength more than mere wear of friction or rupture of fibres : it is a loss of molecular elasticity, and indicates the importance of slight bends, and some form of flexible elastic connection between the socket and the cage or tub. The larger the wheel the greater the durability of the rope. The ratio of their diameters should be as 100 to 1, the minimum drum 48 times that of the rope. A 7-wire rope requires a larger drum than a 19-wire rope of the same diameter. A 1" rope, at 140 lbs. to the 100 feet, good for a working load of 9000 lbs., requires at least a 4-foot drum. It takes a 3" hemp rope to give an equal strength. The weight of a wire rope in pounds per foot is ascertained by multiplying the square of its diameter in inches by 1.58.

As the depth of the shafts increased, it was found that the use of the round rope was attended with many difficulties. Its size and weight increased enormously, the inequality of the work of the engine became more marked, and its drum cumbersome. To obviate these troubles, and the whirling of the bucket as it travels in the shaft, flat ropes and reels were introduced. They are formed by placing side by side and uniting by wire several round wire ropes. The adjoining strands have their wires wound in contrary directions, to counteract any tendency to untwist. The running weight of flat ropes is greater than that of round ropes of like material and equal strength.

There is a limit beyond which a rope of uniform section cannot safely carry its own weight. It will break with 12,000

feet, exclusive of any extraneous load. The additional weight of cage, car, and mineral (say 5000 lbs.), and the energy exerted in starting it, reduce this limit materially; the safe weight is from $\frac{1}{4}$ to $\frac{1}{6}$ of this. By the use of the tapering rope there is no limit within mining possibilities. The lower portion is of an area sufficient for lifting the cage and its contents above it; the rope is graded in size, proportionate to the increasing strain of its own pendent weight. The taper helps to equalize the work of the engine. Practically, a tapered round rope is preferred to a flat, which on occasion slips off the top coils and becomes wedged between the reel-side and the lower coils.

The tapered rope is not well adapted to long slopes, for its lower portion, which is the smaller end, traverses a greater distance than any other portion of the rope and is subjected to the greatest portion of the wear. This is the reverse of the situation in a shaft, where the thickest part of the rope has the wear.

The cables are preserved by smearing with dope, or hot coal-tar. A bushel of lime is added to each barrel of tar, to neutralize the acid and prevent corrosion. Hoist ropes are never galvanized; the zinc soon rubs off, and then electric action is set up, rendering it worse than useless.

27. The life of a rope will depend upon the good condition and adjustment of the wheels and drum, and the care taken against corrosion and undue friction. The coiling and uncoiling of wire rope in the same manner as hemp injures its flexibility and displaces the strands. It is safe for only eighteen months of continuous service. The ropes of tramways do service for a longer period where precautionary measures are not so urgent as with vertical lines. The flat wears out more rapidly than the round, because of the winding on itself in reels. Indeed it is this rapid wear which counterbalances the only advantage that flat ropes have—that of equalizing the engine work,—that militates against its more extensive use. Where a spiral groove is not turned on the drum for the reception of the round rope, or where the drum is near the sheave, the rope is rapidly destroyed by chafing. The most dangerous element

of destruction is vibration set up in the wire by jars, caused by careless starting and stopping, or by jerks from coiling on a drum too small. The paying out of an excess of rope is injurious, because while hauling in the slack the engine gets up speed, which jars the rope when taut. A short chain between the rope-socket and cage forms an elastic connection, and partially corrects this (Fig. 50). As the links wear rapidly they should have frequent examination. Every few weeks they should be annealed by heating to a red and cooling in the air.

Rapid hoisting tends to shorten its life, since the strain imparted to the rope is proportional to the square of the velocity. For a given output, it is better to decrease the velocity and increase the load. This is not always possible; for the live load is fixed by the size of the cars, and that by the hoisting-compartment size, which cannot be altered. The remedy is a large shaft at the outset (see II, 61).

The rope is inspected weekly by feeling for broken wires as it passes through the hands while hoisting. If any short length of it has numerous ruptures, the defective part should be removed. Along a long rope many of the wires may be frayed and broken without condemning it.

There are two types of sockets for round ropes—the conical and the double-pin, the former being stronger. The conical socket (Fig. 51) is slipped on to the rope, the wires are untwisted, hemp centres cut out, the wires bent back and forth into a tangled mat to fill, as nearly as possible, the conical socket, which is then slipped into place. This is slightly heated, and soft lead poured in to solidify the mass. The socket and rope are surrounded with wet clay to prevent heating of the wires beyond. The double-pin is treated in the same manner, but its connection with the chain is by a pair of pins through the links, instead of a ring for hooking, as in the former case. A “goose-neck” socket consists of a pair of trough-shaped tongs.



FIG. 50.



FIG. 51.

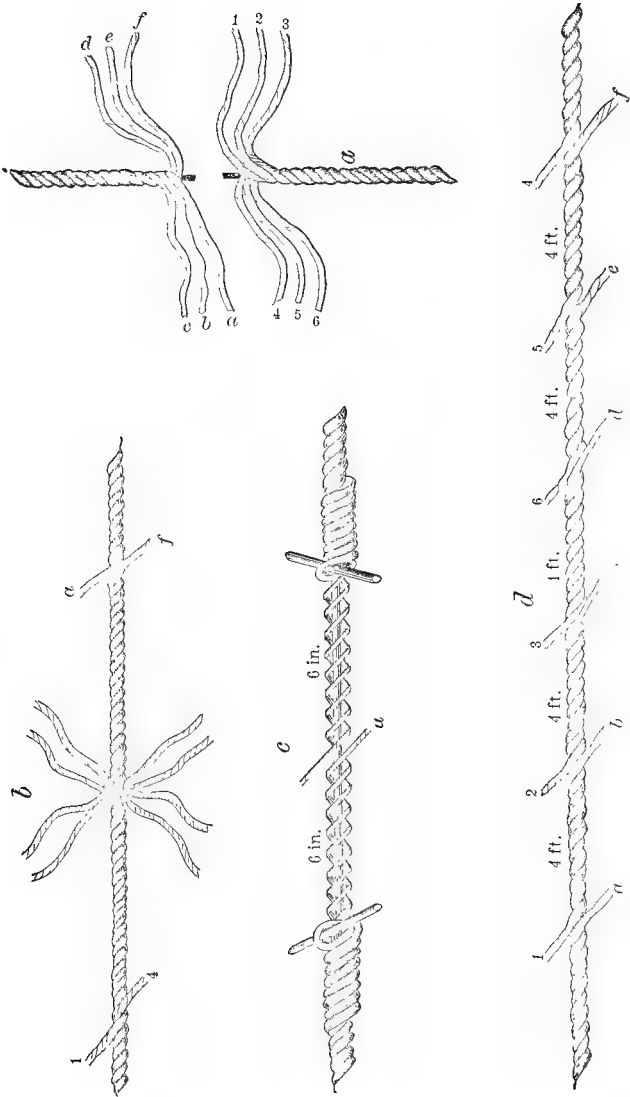


FIG. 54.

bent to a loop, and riveted to the rope by three or four rivets, driven cold. Flat ropes have riveted to them shackles with eyes, which receive the first link of the chain. Six inches of the end are untwisted and doubled back, bound with wire, the shackles slipped on, riveted through the rope, and the hoops finally slipped on and driven tight.

In securing the rope on the drum it is only necessary to continue several extra coils of the rope, insert the end through the wooden lagging, and fasten it on the hub or shaft; or, instead, the end may be bolted to the arm of the casting. If the fastening have but a 10-lb. grip on the rope, it will resist a weight of 90 lbs. if there is only one coil around the drum; if there are two extra coils 800 lbs. will not budge the 10-lb. grip; with three extra coils it requires 7300 lbs.; while with four it has a 65,000-lbs. resistance.

Wire rope is spliced in the same manner as hemp. The strands are unlaidd for 3 feet, and each passed over one and under another of its corresponding strands on the opposite rope, for a like distance; the free ends are then trimmed off close. Short twisted rope is more easily spliced than one of long twist. The average cost for rope per ton raised 100 feet of shaft is 0.053c., and 0.069c. per 100 feet of slope. From a report (1884. of the German Government), the cost of winding is as follows: charcoal iron, round, 0.04c.; steel, round, 0.06c.; iron, flat, 0.08c.; steel, flat, 0.14c.; aloë, flat, 0.13c. The relative economy in the use of hemp and wire rope is still in question among engineers of Continental Europe, many having abandoned iron for hemp, claiming the former to be 80 per cent dearer. Their conclusions of practical tests are evinced in the almost universal adoption of aloë and hemp ropes. Their estimates show that these ropes carry equal loads for equal weights with wire. The mode of splicing wire ropes is illustrated in Fig. 52.

The following citations are made for the student:

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Amer. Inst. M. E.: Dumping-cradles for Mine-cars, H. S. Munroe, XVII. 564.

Ill. Min. Inst.: Safety-catch, II. 109, 159, 180, 212.

Coll. Guard.: Stops for Mine-cages, B. Wilmotte, 1896, 1173; C. S. Smith, Dec. 11, 1896, 1123; B. Schiechel, Jan. 1897, 33; Spring Couplings, Feb. 1897, 317.

Colo. S. of M. Quart.: Mine-ropes, strength, care, etc., C. Aquillon, I, 1892, 20.

School of Mines Quarterly: An Ore-bucket for Inclined Shafts, Alexander L. Black, XV. 47.

E. & M. Jour.: Wire-rope Tramway, English Mt. Mine, Cal., LIX. 55.

CHAPTER VIII.

UNDERGROUND TRAFFIC.

28. Description of cars, low *vs.* high; investigation into the minutiae of rolling-stock; wheels and self-oilers; gauge and grade; spragging; automatic devices against runaways. 29. Life of a car, dumping cradles, etc.; rails and turnplates; economy of rolling ways; consideration of friction, grade, consumption of power, etc.; tramming by hand; work of a man or animal in haulage; mules and horses, their cost and efficiency, compared with mechanical appliances; grades and the various limitations to haulage powers; objections to underground engines. 30. Locomotives for underground haulage; their sizes, speed, cost, and efficiency; smokeless, pneumatic, and electric engines; details of gravity roads, self-acting inclined planes, engine planes; clips, wheels, brakes. 31. Tail-rope systems; details, size, and cost of plant; mode of passing around curves. 32. Endless cable systems; descriptions of the four varieties; comparison of their advantages and adaptability; report of the tail-rope committee; example.

28. FOR the transportation of mineral along nearly level roads, from the working face to the entry, and on cages of carriages up the shaft or slope to the surface, various forms or cars are employed. Their size is determined by the dimensions of the gangway, and the demands of cheap haulage. Evidently the car should be as light as compatible with strength, and such that a rammer can easily manage it. Tightness is an equal requisite. For this reason a boiler-iron body is preferred, though not so much in collieries, where the wooden box is used (Fig. 53). The weights of cars vary widely—usually about half that of their cubic contents. In metal mines they weigh from 600 to 1500 lbs., and their cost is from \$50 to \$200. Some colliery cars carry as much as 120 cu. ft. of mineral, those in metal mines not over 30 cu. ft. Undoubtedly haulage in

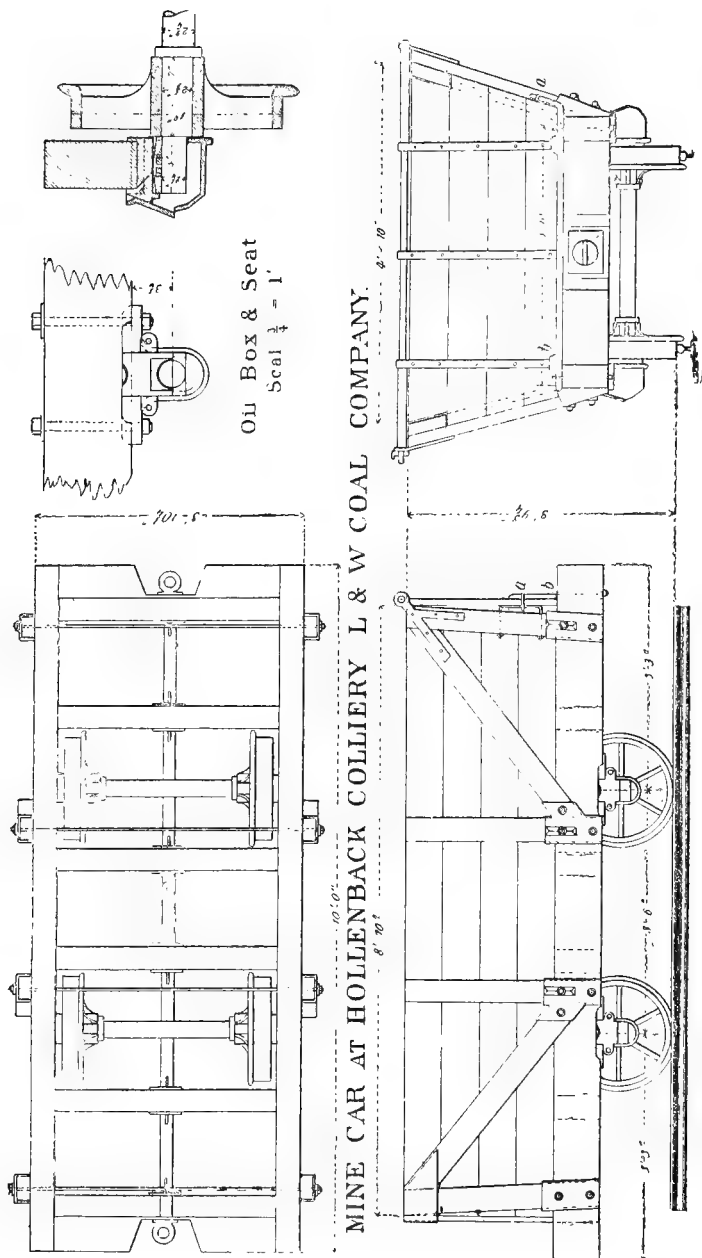


Fig. 53.

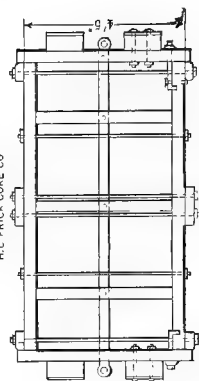
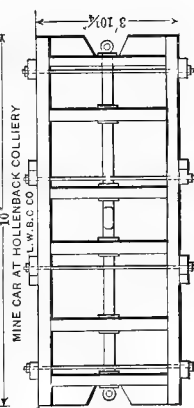
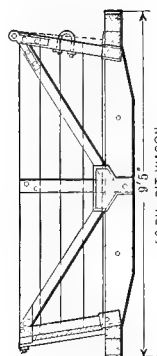
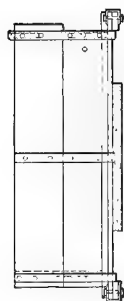
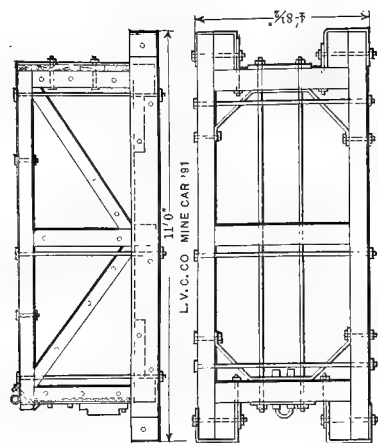
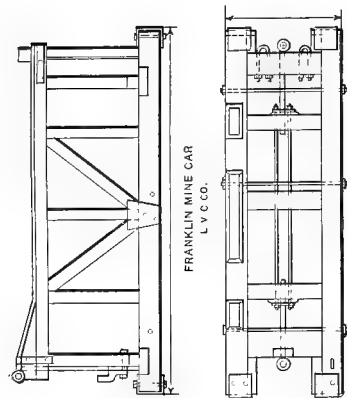
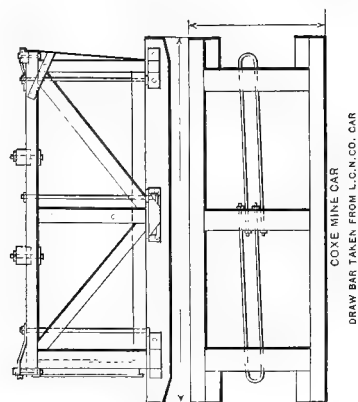
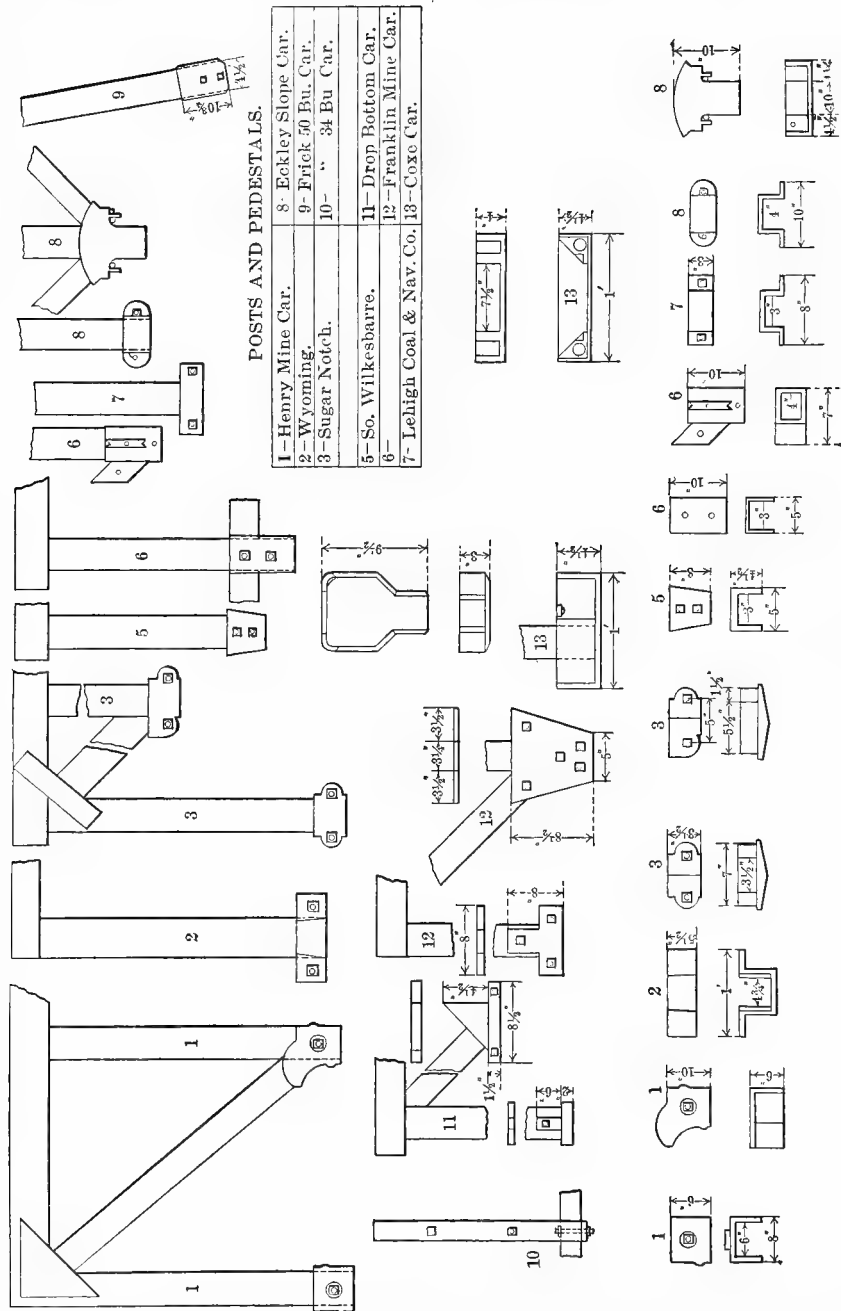


FIG. 54.—WAGON-BOXES.



POSTS AND PEDESTALS.

1—Henry Mine Car.	8—Eckley Slope Car.
2—Wyoming.	9—Erie 50 Bu. Car.
3—Sugar Notch.	10— " 34 Bu Car.
5—So. Wilkesbarre.	11—Drop Bottom Car.
6—	12—Franklin Mine Car.
7—Lehigh Coal & Nav. Co.	13—Coxo Car.

FIG. 55.—CAR-BOX IRONS.

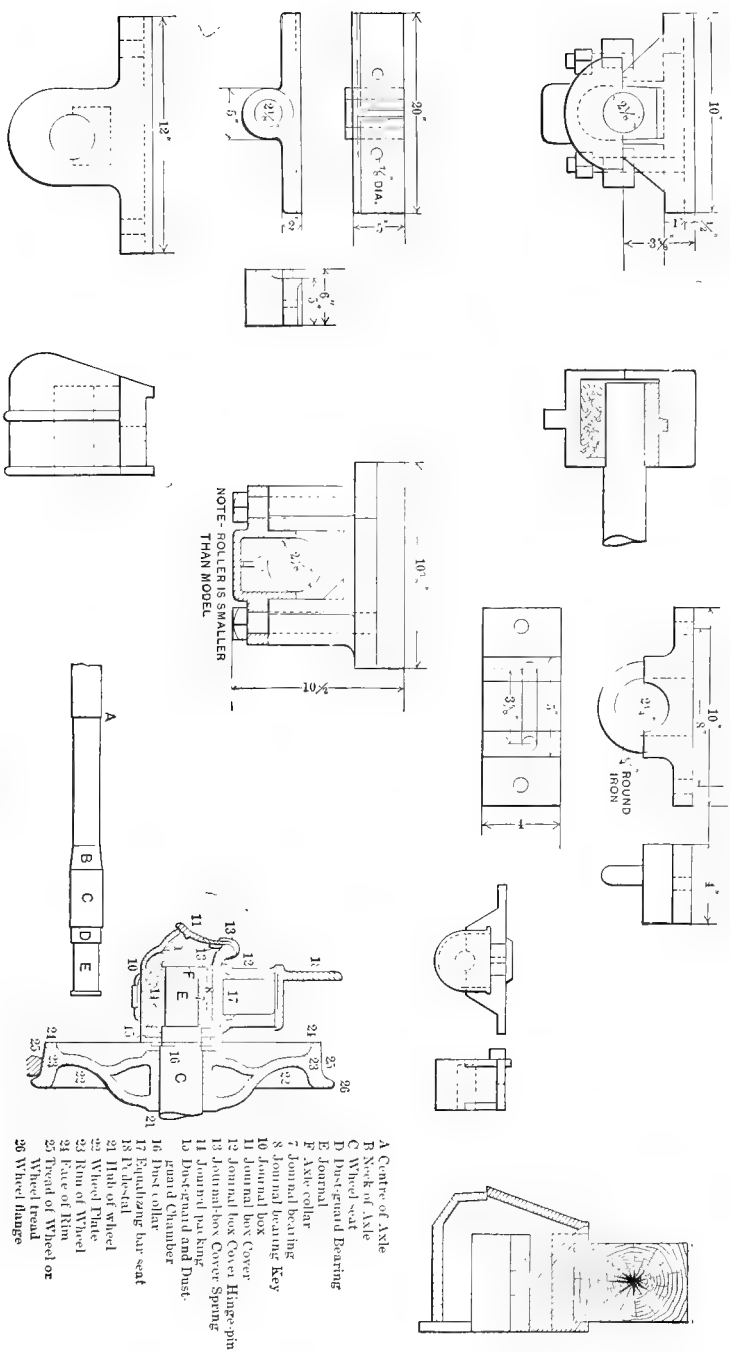


FIG. 56.—DETAILS OF JOURNAL BOXES.

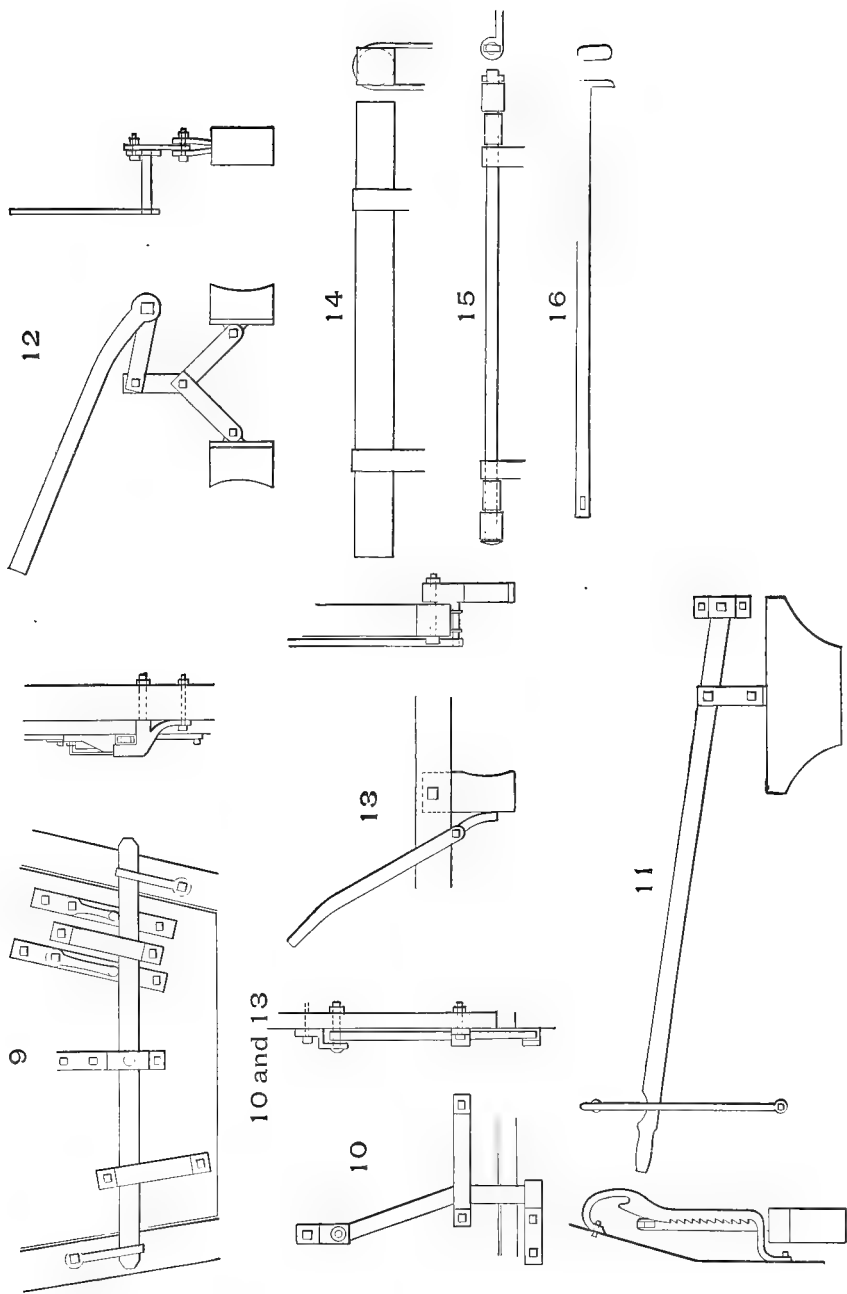


FIG. 57.—LATCHES FOR MINE CARS.

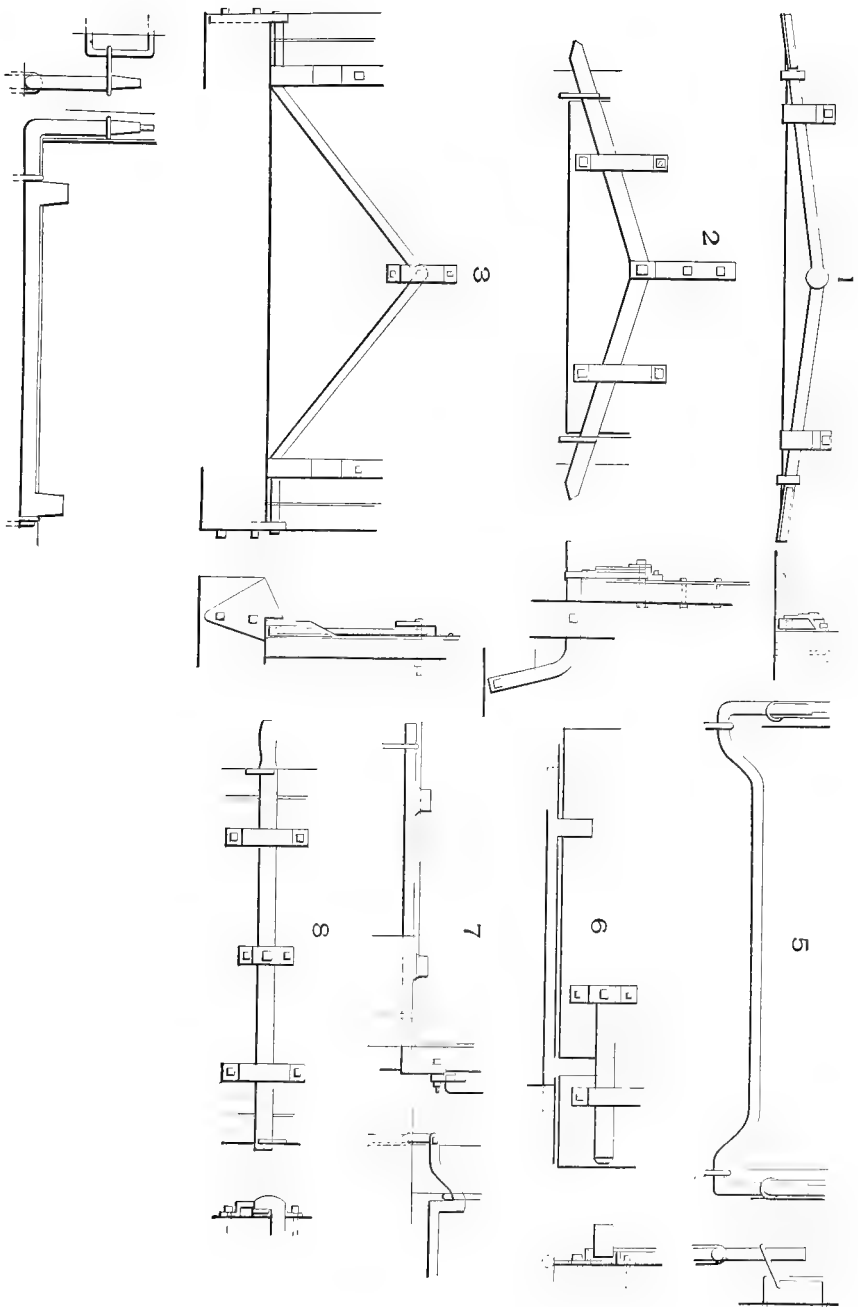


FIG. 58.—DETAILS OF LATCHES.

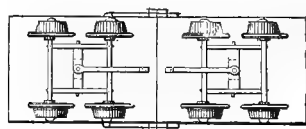
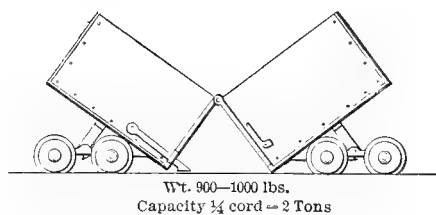
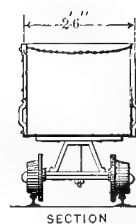
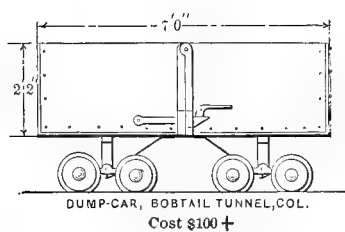


FIG. 59.—TUNNEL DUMP-CAR.

large cars is cheaper, so the tendency is toward a great capacity, though the natural conditions of underground work restrict the dimensions. The small cars used in buggy roads (see p. 34) have a capacity of 26 cu. ft., and are made for a 3' or 4' gauge. The length of the cars is limited by the sharpness of the curves; in collieries the maximum is 9 feet; in metal mines it is 70 inches. Their height depends upon the conditions of loading. In thin seams and steep veins cars only run in main haulways, and are filled from chutes, provided with a spout and gate, easily manipulated at the bottom; if also hoisted on a cage, their height is a matter of indifference. When the seam is thick and the roof good, they are carried up to the face of the work, in which case they are filled by hand. If so, or if raised on carriage, the difficulties and expense are greater with a high car. To shovel one ton into a 3-foot car requires over 7300 ft.-lbs.; into a car 4 feet high, 9500. The average man can exert a continuous shovelling effect of 28,100 ft.-lbs. per hour. Allowing for the weight of the shovel, delays, throwing the mineral forward, a shoveller may load about 20 and 14 tons, respectively, in the cars per shift. Even for a medium output the economy is manifest. In metalliferous mines this is observed, but in collieries cars of 4' 9" and over are common. For stability, too, a low car is desirable. The width of the car depends upon the gauge and its "set." Broad cars are preferable, but may not be advantageous because of the wide gauge. Nor are they desirable if set up on a narrow gauge. A compromise is frequently taken, by which a low, wide car on a narrow gauge is employed. The axles are elbowed for large wheels, and set down on them is a narrow body, which bellies out wider over the wheels. All mine cars should be provided with bumpers, to keep the bodies of said cars at least 12" apart.

The gauge varies from 2 feet to 4 feet, with good and sufficient reasons for the choice of any intermediate. Broad gauge gives greater stability, and a reduction of haulage-expenses. The minimum gauge of 2 feet is advantageous for easy haulage and sharp curves, cheaper track and rolling-stock, but

tends to reduce stability and capacity. It reduces the length of the car, but allows of the use of inside wheels.

The wheels are as large as circumstances will permit (the larger the wheels and the smaller the axles, the less is the friction). The wheels may revolve loosely on the round or the square axle, or they may be fixed to the axle and revolve with it. Some are capped with a recess in the hub, to receive the collar on the axle, and thus prevent admission of grit (Fig. 56). They may be "inside" (below), or "outside" (beyond), the body of the car. As to the relative merits of the inside and outside, or loose, wheels, it must be admitted that engineers are not united in the opinion, though the former has the larger number of adherents. Outside wheels are more easily oiled, are cheaper and admit of the body of the car being set lower down; they do not run so smooth, or last as long as those fixed under the body of the car. Loose-wheel cars may be better for short roads with sharp curves, but they are harder to pull. With fixed wheels, one of the mutually dependent wheels, in travelling about curves, must slide. For this reason, and the ease of lubrication, loose wheels, or cone-fixed wheels, are preferred by many. The U. P. R. R., 3 feet gauge, abandoned loose wheels after careful trial. At the Drifton anthracite mine, a compromise is effected by using a pair of fixed and a pair of loose wheels.

The coal-car wheels are of cast-iron, between 16 inches and 18 inches diameter, and those of ore-cars about one half that, and solid; while the former have hub and arms to allow of "spragging."

Coal-cars are fixed on two trucks, and dump from the end, being provided with a swinging door (Fig. 54). Iron-frame cars are more commonly provided with a swivel and a lever, which hooks or unhooks the body from the trucks (Fig. 60). Dumping is easily effected by opening at the side or end a swinging door hung on an iron rod across the top by two hinges (Figs. 61, 62, and 63). Another variety of car in use in tunnels consists of a double iron-framed car, pivoted together at the centre top of each side, on two trucks. In dumping, the latch-lock on

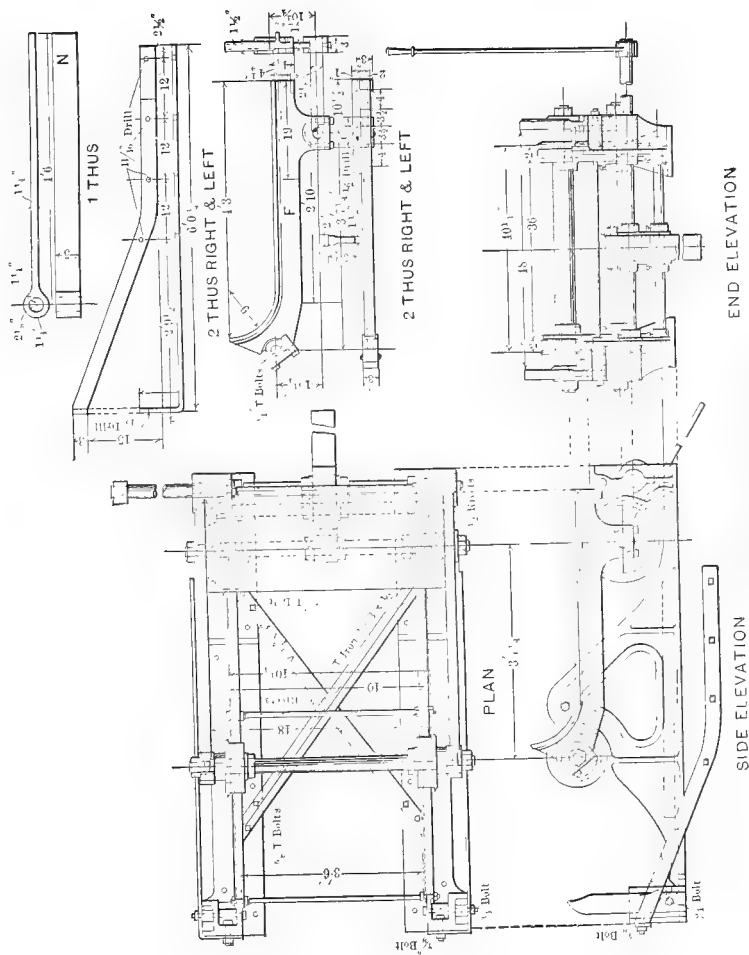


FIG. 63.—ANTHRACITE (continued).

each side is raised, the car opens in the middle and empties between the trucks (Fig. 59).

Often the dumping is accomplished at the surface by some

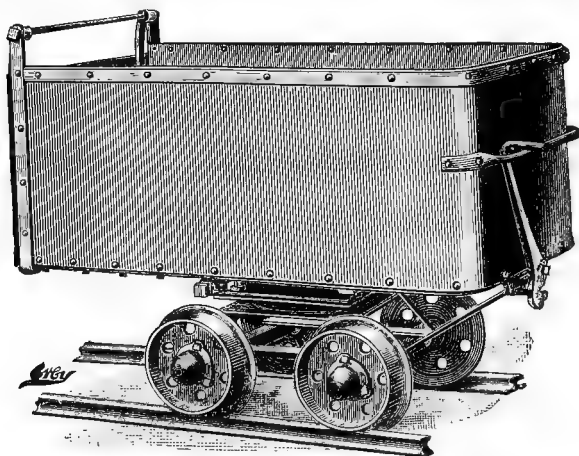


FIG. 60.

automatic device, consisting of a balanced frame, or pivoted cradle, upon which the loaded car is run and held. The moment

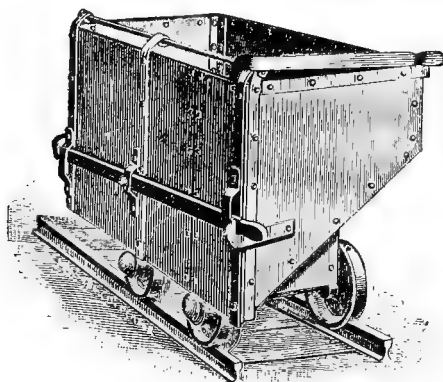


FIG. 61.

its centre of gravity is beyond the point of support, it tips and empties the car. In another type of the dumping cradle, the entire combination is inverted; the car, which is held, dis-

charging its load at once. Behr's device is of this character and is represented in Fig. 65. This permits the use of a stronger and lighter car, and dispenses with a dumping device on the car. A similar idea is adopted in the construction of the cars, without the use of the cradle. The body of the car is hung on a horizontal axle over the truck, so that the centre of gravity of its contents is very near to the fulcrum; a very

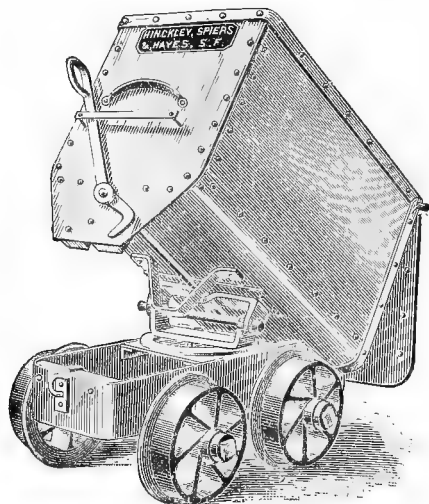


FIG. 65.

small effort by the trammer will tip the car after it is unlocked.

As much attention should be given to even the most insignificant details as on surface roads, to secure maximum economy; rigorous examination into anti-friction method is advised. Bearings and axles should be readily accessible. Self-oilers can be had, simple and cheap. They save power and reduce the wear. By their use, one-third of a pint of oil will last two to four months, according to the quality and the distance traversed. Their addition, in one mine, saved \$7 per year per car, in oil and grease. Another example, as a corroboration, taken from an ordinary trip, showed that a locomotive can haul 20 cars with plain wheels, with a loss of 12 pounds steam-pressure, against 28 cars with Bowden self-oilers and a loss of only 3

pounds. The oils used for lubrication and illumination, underground, in the Lake Superior mines, are let down into the mine through a small pipe into tanks, instead of lowering the barrels.

The haulage inclinations and velocity being slight, brakes

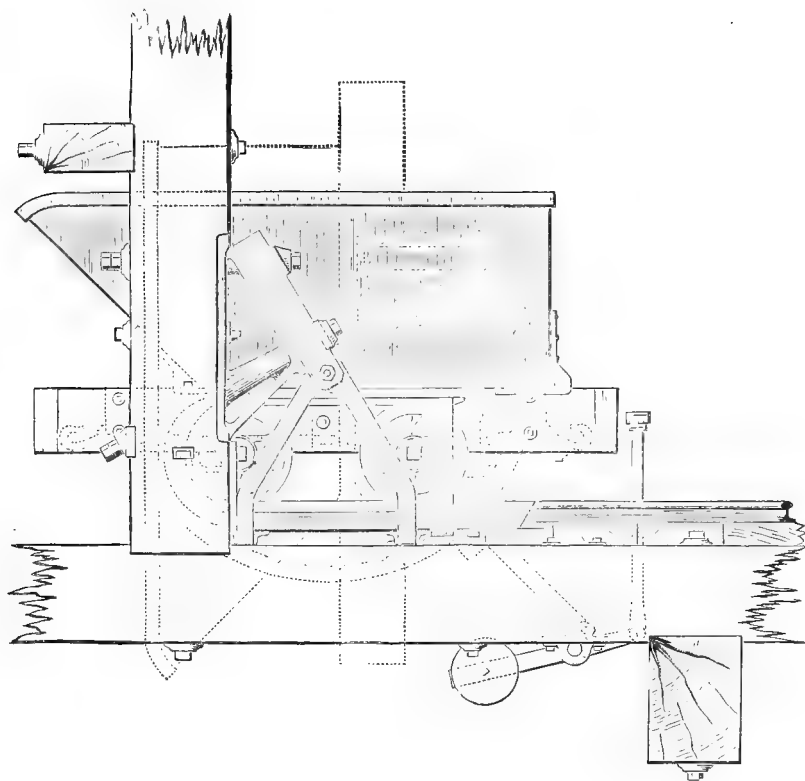


FIG. 65.

are not needed, nor are they much used, unless the stout stick with its fulcrum under the body of the car, and pressed on the wheel face by the weight of the trammer at the other end, may be dignified by the name. In coal mines, "spragging" is resorted to, and is more effective. A sprag is a billet of wood 12" long and 2" thick, which is deftly thrown between the arms

of the wheels and prevents them from turning, and converts the car, more or less, into a sled. On a slight grade only one sprag may be required, while on a slope of 1 in 6 it takes 4 sprags to check the speed of the car. They are of no use on an incline of over 1 in 5, the angle of sliding friction. Roads are often designated as 1-, 2-, 3- or 4-sprag roads, by which is meant the number of wheels spragged.

To prevent accidents from runaway cars on a grade, or from the mill back into the shaft, automatic devices are used. They are usually a balanced timber block automatically thrown across the track before the car reaches it, temporarily blocking the way. See Fig. 40.

The life of a car depends upon conditions too varied to state here. Wooden cars become loose, shaky, and larger with age. Iron ones are battered and bulged, particularly if the mineral is in large lumps.

29. The tramway is of T rails, weighing in rooms, 12 pounds, in gangways and levels 16, and in slopes as high as 35, per yard. Very light rails are not economical. A broad rail favors the wheels; depth and weight give stability. Only in petty mines does the strap rail survive, and in some steep slopes the wooden strap. The cost of 1 mile of 16-pound rail, laid, is \$1600 (steel at \$33). If the floor of the gangway is uneven, the sills or sleepers are laid on the knolls or ridges. For heavy duty the gangway is double tracked, or, sometimes, a single track is laid with suitable turnouts and plain frogs. At junctions, a simple iron turn-plate laid on stout planks is used, instead of the more elaborate frogs and crossings. The men drag the end of the car around, and shift it to the desired track, when it is run off. Self-acting switches are not in favor.

The opportunities for economy in underground work are not many, for the conditions are necessarily proscribed. Mechanical appliances are difficult of application, and particularly so in veins of high pitch, where the inconveniences increase, because of the narrowness of the gangway. Other lines of economy are easily obtained, and the virtue of well-laid track specially commended. On level surface roads, the trac-

tive force required to overcome friction is about 1 per cent of the load. Underground it is rarely less than 3 per cent. R. Van A. Norris, Wilkesbarre, conducted an elaborate series of experiments, with the result that the coefficient of friction was rarely less than 60 pounds per ton, occasionally 100 pounds is reached. With self-oilers this was reduced to $1\frac{1}{2}$ per cent. With any mode of rope haul, the frictional loss of power may and often does amount to 40 per cent of the weight. There is no reason why the same care should not be employed below as above ground.

The difference, in the cost and time of laying, between a substantial track and one poorly laid is trifling, but in efficiency is astounding. One car running two-thirds of the year consumes, per ton of mineral it carries, on the latter, an equivalent of 30 tons of fuel more than it would on a decently laid road, and the amount that an animal or a locomotive can pull is reduced to a quarter of that on a good road. Perhaps it may not be regarded as essential; it is nevertheless advisable that good workmanship be expended upon the trackway and rolling-stock. This is more urgent as the output is large or its value small.

For a small turnout, or on a short level haul, man power may be employed; but, as any of these increase, power must be invoked. For steady tramming, the average man is capable of exerting 27 pounds push, at 2 feet per second, moving a 2-ton car on a level, and making, according to the condition of the roads and the running-gear, from 3 to 12 ton round-trips of one mile.

Underlike conditions a horse or mule makes three single trips with from 4 to 10 tons gross load per hour; though the delays, friction and bad air reduce the average to 40 or 50 gross ton-miles per day at 4 or 5 cents. The utility of the animals is confined to haulage in the secondary ways, the rooms, and for switching-places where economy in height is practised. This gives the mule a superiority over horses, because it is not so tall, though equally strong. Even as it is, the roof must be ripped off from beds below the standard thickness, to admit of

a gangway high enough for mules; often, dogs or pushers are employed instead.

Usually mules are driven in teams of from two to six, according to the length of the trip and of the train, averaging two cars to the mule. Their ordinary speed is about two miles per hour. Some small mines employ them for haulage from working breasts, or only from the secondary ways, to daylight. In larger properties the mule never again sees daylight, and travels between the rooms and the general parting, or even only in the branch roads. Except where there are numerous ventilating doors to be opened and closed, or many sharp turns to be rounded, there are few cases in which the mule has the advantage over mechanical means, and where the railway may not be extended throughout the entire workings without great additional expense. It is the privilege of the operator to replace animal power by machinery, and just in proportion as he avails himself of the latter, so will his profits be.

The average cost of mules is about the same as with horses. It takes thirteen animals to supply ten workers, the balance being on the retired list for various causes. This excludes the allowances for death by accident. The cost of their keep is 90 cents per day. The accommodations for stabling need not be expensive, but attention must be given to ventilation and cleanliness. Under average conditions one mule will serve for the haulage of an output from ten miners.

About 9000 gross ton-miles per year represents the work of one animal, which, however, varies with the grade and condition of the track. The limit is determined by the tonnage that an animal can return with uphill. Only a small grade is admissible, and that, too, from the breast. The maximum is somewhere near 3 feet per 100. A gradient of equal resistance (that on which the work on a loaded car down equals that on the returning car) should be provided where possible. If the empty returns with stowage and supplies, the grade of its track should flatten as their weight approaches that of the mineral, as, otherwise, the duty of the animal is lowered appreciably. A power which will pull 100 tons on a level can take only 47 up

so slight a grade as 20 feet per mile, and 25 and 13.5 tons respectively on a 1 per cent and 2 per cent incline.

It is easily understood that the development of an extensive property by numerous shafts is expensive, so it is likely preferable to transport the mineral underground even a considerable distance to a shaft centrally located, if this can be done quickly and cheaply. This involves the elaboration of a system of haulage depending upon conditions of grade and the method of mining, as follows:

1. Where the tramway is horizontal, power is required both ways by man, horse, locomotive, stationary engine or rope-ways.
2. If its grade is towards the shaft, the full train down pulls an empty up, on a self-acting plane or tramway.
3. When the grade is reversed, the loaded and empty cars are moved by a tail-rope or endless-rope arrangement.

The method adopted should not be complex, all the details carefully proportioned from direct calculations, the positions of branches and the location of machinery comprehensively planned; often the opportunity for obtaining cheap haulage fixes the entire plan of the mine.

The advantages offered by either of these systems cannot be generally stated. Any plan allowing of a single track in a narrow gangway usually has the preference, though on a double way no extra power is required to overcome the friction and the dead-weight of the conveyance. Of course the most favorable tramroad is that affordnig a down-grade on each track. But as this is not feasible, it behooves the engineer to diminish the frictional resistances and avail himself, as much as possible, of the acceleration of gravity. Power other than gravity is indispensable when the grade is low or against the loaded car. Stationary engines cannot be employed underground because of the dangers and inconvenience from sparks and exhaust steam, unless air be the motor fluid. Steam-, pneumatic-, or electric-locomotive or rope traction is then necessary.

30. The locomotive furnishes a cheap haul for great distances and large output (Fig. 66). Its simplicity and convenience recommend it to favor; it is much cheaper than animal power, and has the advantage over it in times of strikes and lockouts (it has not to be fed); it is easily accommodated to varying demands on it. As it is usually coal-burning, the gaseous products turned off into the mine justify the outcry against it. It befouls the ventilation, introduces a risk from fire, and also elements favorable to the decay of the roof and timbers,—heat and moisture. Its passage up and down interferes markedly with the volume of air traversing the entry; when the locomotive travels with the current, 20 per cent more air passes

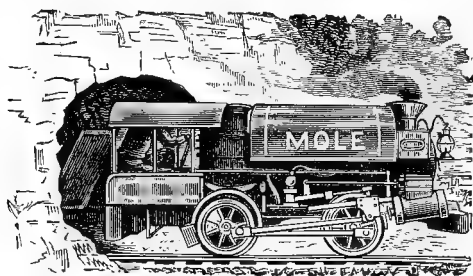


FIG. 66.

through than while travelling against the current. On this account the ventilating current should have a greater speed than the locomotive. However, its gangway is usually cut off from the general ventilation of the mine, the inlet current being introduced beyond the inside terminus of the locomotive run. The cost of these improvements, preparatory to this introduction, is no small item (\$7000 in one mine).

Still, to a large extent, much the same objections obtain to any underground steam-engine, compared with which its greater haulage velocity results in less cars for equal tonnage, and less cost (2 cents per ton-mile). On the other hand, it is useless on grades of over 3.5 per cent.

Compressed air may be used for the power, instead of steam, and afterwards for ventilation. If this is done, station-

ary engines will have the advantage over locomotives ; otherwise, all things considered, the latter will ordinarily be preferred ; and it is remarkable what dry steam they furnish, and what work they accomplish considering that their draught height is limited, the rails wet, and the curves sharp.

The locomotives are made of a shape to suit the mine opening, for narrow gauge (36'' to 40'') rarely over 78'' high, have four to six wheels (for curves of 50 to 75 feet radius), weigh 4 to 13 tons, and carry 125 to 350 gallons of water. Their cylinders are from 5×10 to 10×14, on 22'' to 28'' drivers, running over 16 to 28 pound rails, and costing \$2,600 to \$4,000. They have a traction of from 150 to 600 tons on a level. There is no difference in the price between the wide and narrow gauge locomotive of the same design and size of cylinders.

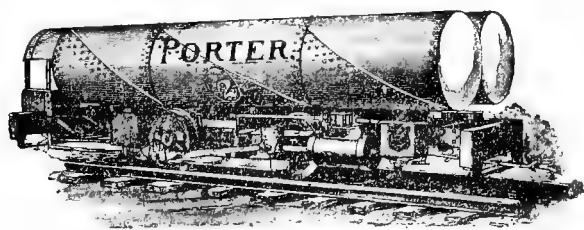


FIG 67.

The hauling capacity (the total weight of train guaranteed to be hauled on a level, straight track) is limited by the adhesion of the drivers to about one third the weight of the locomotive. A locomotive with a pair of cylinders 6'' × 10'' will, on a grade of 105 feet to the mile, haul 28 tons of train 20 miles daily on 600 pounds of coal. One of 10×14, on a 52-foot grade and 50° curves, has an actual duty of 46 tons, 28 miles per day, with 1000 pounds of fuel. The average grade in mines is about 2 per cent, on which the capacity of the locomotive is 13 per cent of that on the level. Grades are usually reduced on curves 0.02'' per 100 feet for each degree of curvature. The daily running expense of a locomotive is \$4.50. Locomotives with inside cylinders are advised for narrow tunnels only.

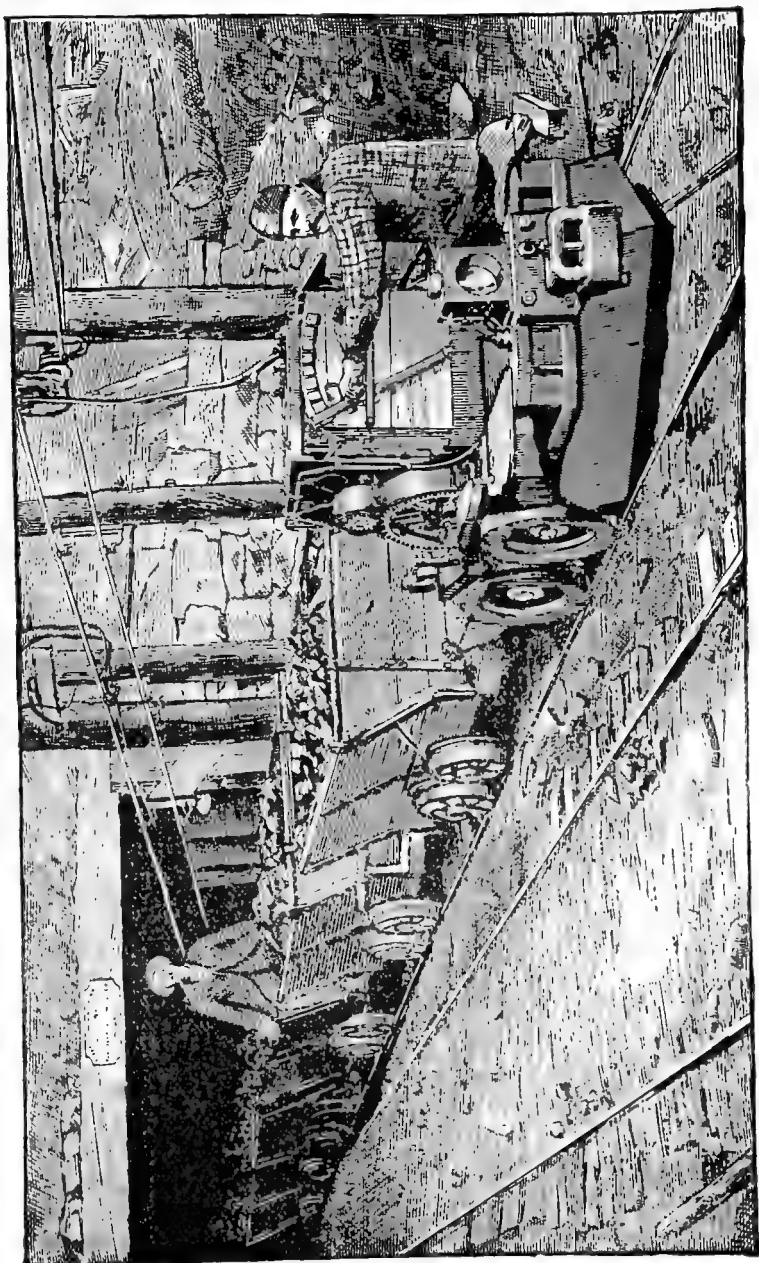


FIG. 63.

The tractive force, T , of a locomotive is measured by the formula $DT = 0.0654k^2ps$, wherein D is the diameter of the driver in feet, and k , p , and s as in 23. The traction of a locomotive, a static force, expressed in pounds, must not be confused with its horse-power, which is a unit of dynamic force, embracing the elements of weight, distance, and time. (Compare with 23.) The traction, T , must be equal to or greater than the sum of the train resistances; 1, the frictional, which in mines is not less than 50 lbs. per ton of train, and equals $0.025Y$; Y is the number of tons weight of train and load; 2, due to grade, which is $20gY$, g being in feet per 100; and 3, due to the curve, $\frac{1}{2}wzD^\circ$; this is $\frac{1}{2}$ lb. per ton per foot width of gauge, z , per 1° curvature; w is the weight of the number of cars which are on the curve at the same time; for substitution, when the radius of curve, instead of its degree, is known, we have $D^\circ r = 5,730$.

Pneumatic locomotives (Fig. 67) are not yet successful, nor the various fireless and smokeless engines constructed to be operated by volatile chemicals, leaving electricity and the wire-rope systems as the only real competitors of steam-locomotives, which are confessedly not as economic or as safe machines as stationary engines.

The signal success of electric installation has led to improvements in haulage methods by the use of motors (Fig. 68) and storage-battery. Aside from other considerations, the rotary movement of electric appliances admits of a better running balance than can be given to ordinary reciprocating engines, and they are therefore less liable to jump the track. An 80-ampere, 450 volts Compton series wound machine, operating haulage engines, hauls from three parts of the mine, by cable 3200 yards long, 100 tons per day, and replaces 27 mules, besides several helpers, etc. As yet electric propulsion is more expensive than steam, and gives more trouble, because of the liability to indefinite delay from frequent groundings and other mishaps peculiar to electricity when carried by trolley-wire as in the surface roads. Judging from the Census Reports, the interest on the big plant is a very large item in the cost of electric propulsion. The current is carried by a No. 0 wire, T- or L-irons being used for the trolley.

A loss of fifteen per cent is usually allowed for the line loss in voltage from the generator to the end of the run, and an efficiency of eighty per cent is estimated for the motor.

Storage batteries obviate the necessity for wires, but are as yet too expensive. About 25 lbs. of battery will carry one horse-power per hour. A space of 40 square feet will accommodate 250 elements, which will furnish 11 horse-power for a 10-hour shift. The weight, and hence the adhesion, of this engine is over twice that of the steam locomotive of equal horse-power, even if the coefficient of adhesion is not increased.

Where the conditions are favorable to the use of gravity as the sole power, on a self-acting plane, the principle employed is to let a loaded car going down pull an empty car up. A rope connecting the two cars, passing around a sheave, or a drum, at the head of the incline, is the only mechanism required. With the former a single rope is used, while with the latter two ropes, wound in opposite directions, connect the cars. The axis of the drum is horizontal; of the sheave, vertical. The ropes are of a length equal to that of the plane. Often on a drum a single rope is used, which receives four or five extra wraps to prevent slipping, the ends being attached to the empty and loaded cars, respectively. This is not recommended.

The sheave or drum is in a recess fitted for it on or above the counter gangway, where are received and connected the cars from the breast, or sometimes this arrangement delivers direct from the breast; the terminus is on the lower gangway or at the foot of the shaft. Swinging platforms connect the drifts or gangways with the slope. Usually two lines of rails (35 lbs.) are laid the whole length; single tracks with turnouts are false economy.

The smallest gradient at which these are operating is 6° ; the best, 1 in 5; while on one over 1 in 3 there is an excess of tractive force above resistance. At an angle exceeding 35° the method of mining is such as would not afford an opportunity for this means of delivery. Occasionally, in steep veins, such a scheme is in operation for delivery down an "auxiliary" from the several levels to the main gangway. The surplus force must be counteracted by a strong brake, which regulates the speed to a nicety. The mean velocity is about 400 feet per

minute, and to diminish the momentum toward the bottom the plane is flattened. The theoretical curve of the slope is a cycloid, concave upwards.

The brake is usually an iron strap with wooden blocks, actuated by a lever. India-rubber has many advantages over wood, though hitherto the trouble has been to fix the rubber on the shoe, because it disintegrates so readily where the bolts pass through. This has recently been overcome; the rubber blocks have a dovetail at the back, which is inserted and fits easily into its shoe. The tightening of the blocks on the wheel while "braking" crowds the dovetail into the shoe.

Means must be devised for preventing the rope from slipping off the sheave, and provision also made to protect it from undue strains caused by shocks. A V friction-clip wheel, designed so that its friction will equal that on the road, is the simplest plan. The clip bites the rope, and there is no slip or wear unless a car jumps the track, when the slipping of the clutch will notify the brakeman. This clip-pulley is also very effective on endless-rope systems for transmitting power.

The limit of length to which these planes may be used is fixed only by the friction of the dragging rope. Anti-friction rollers, 6×20 inches long, on 1-inch axles, are therefore necessary at intermediate points, about 20 feet apart, to give it support. The amount handled is limited only by the facilities and conveniences affecting the trip time. The system is inexpensive, requires strict discipline and an ample signalling code. The cars may be connected singly or in trains, but equally on each branch. Trailing-forks behind the cars prevent catastrophe if the rope breaks—usually it is the up-rope.

These tramways are equally good underground—as on the surface—where the cost does not exceed 10 cents per mile-ton, and is often as low as 3 cents (Fig. 69). The cost of construction is about \$3 per foot of length. Smaller sized ropes are needed than for an equal length and weight of vertical hoist. Two-car trips on 10° planes require $\frac{5}{8}$ rope, and $\frac{7}{8}$ on 45° slope.

When the slope cannot be self-acting, "engine-planes" are used. A stationary engine is located at the head, and has a

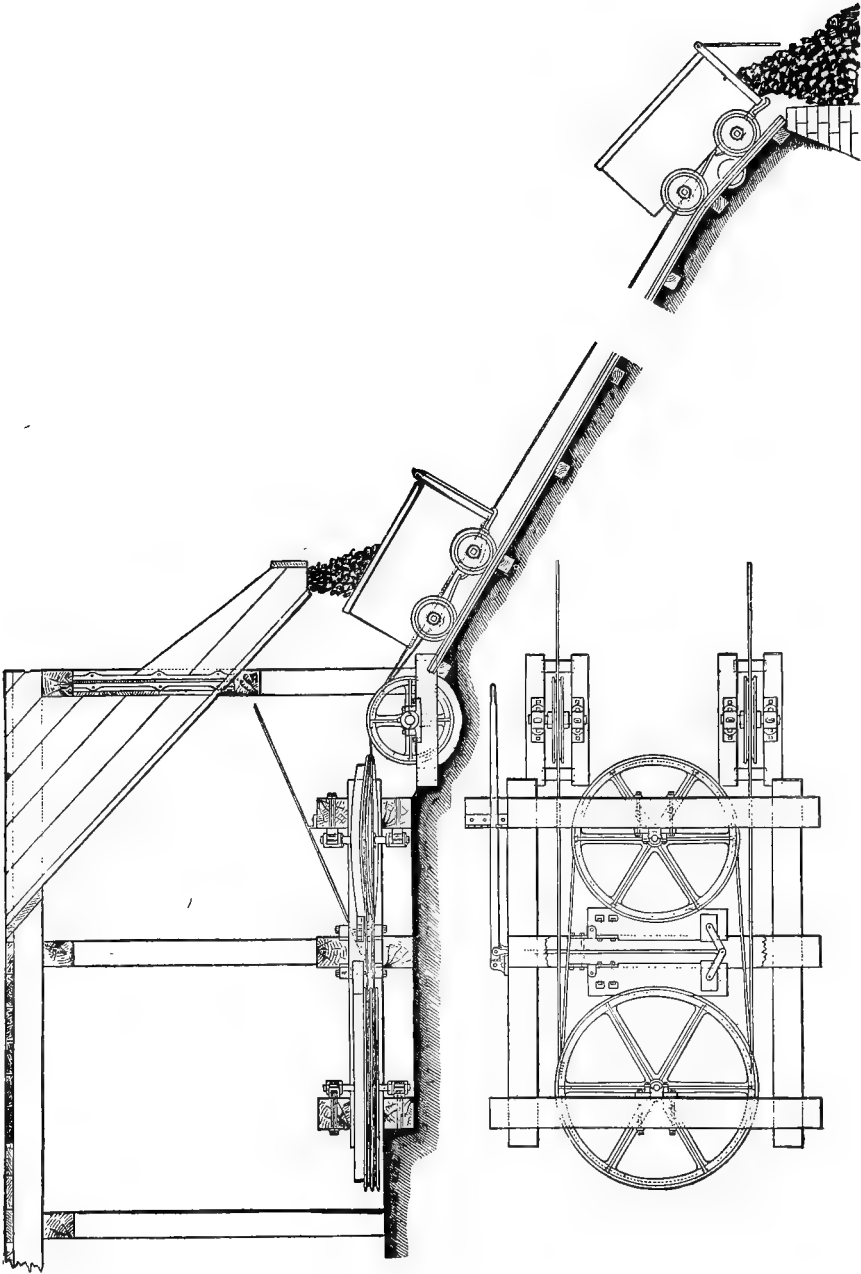


FIG. 69.—INCLINE GRAVITY TRAMWAY.

drum which may freely turn and pay out the rope for the descending cars, and be geared to pull them up returning on the same or a parallel track. On single-track planes the engine is non-reversing. On a grade of 1.7 per 100, gravity will take the loaded cars down with a reasonable velocity (empties on a 2.25 grade) and pull the rope after them. On a 10 per cent grade a break will be necessary.

The cars usually travel in trains, of 10 to 30 in number, in charge of a conductor who operates a dead-fall timber-block to hold the train while the cars are being shunted. So the system is well adapted for delivery from side-entries at different levels, and may be used on slight curves by curving the rope on iron guide-wheels. Ordinarily the rope will last four years. A 14×30 cylinder, 3-ton fly-wheel, $\frac{9}{16}$ steel rope, on a plane 4600 feet long by 80 high, has a daily output of 950 tons of mineral in 25- to 30-car trains.

31. Of rope-ways there are two classes, the tail-rope and the endless cable. Each system has its advocates; both are extensively used in beds, the former for a limiting gradient of 3 in 100, either with or against the loaded cars.

In the tail-rope system, haulage is effected by a stationary engine, two ropes on drums which are thrown alternately in and out of gear. The main rope, having a length equal to that of the road, is hitched to the front end of a train, the tail-rope, of double this length, passes from its drum around the sheave at the bottom, and thence to the rear end of the cars; a short chain at each end of the train couples the cars to the two ropes. In operating, the main-rope drum is thrown into gear, the other out of gear, engine started, and the loaded cars are drawn from the mine to the outlet, dragging the tail-rope after them. Then the main-drum is released and the tail-drum engaged, the empty replace the full cars and return, pulling the main-rope off its drum. See Fig. 70.

The lower sheave is a clip-clutch. The main-rope is rarely over $\frac{3}{4}$ inch diameter. It is generally replaced every year by a new one, doing two years of additional service as a tail-rope, after which it is discarded.

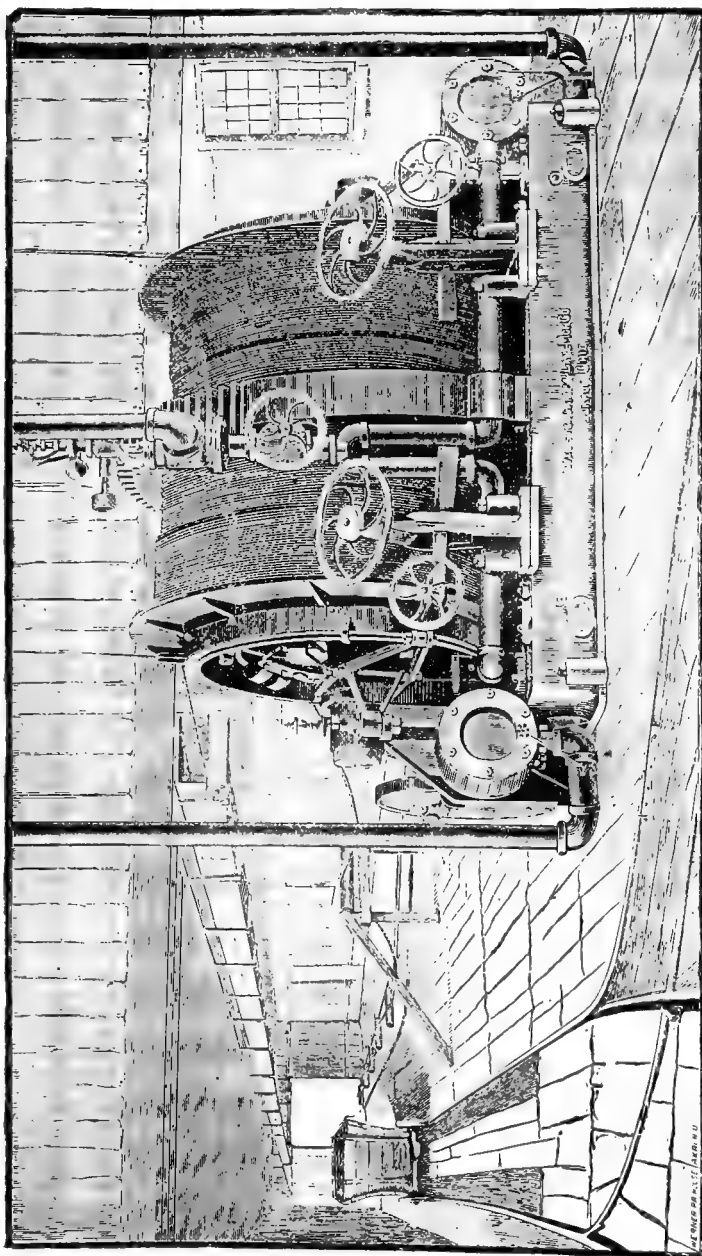


FIG. 70

The length of haul is limited only by the engine-power and the resistances. While the dip may be anything less than 3 in 100, yet its greatest advantage is manifest on a level or very slightly falling dip. The velocity of haul may reach as high as 10 miles an hour. Each trip takes a train of 10 to 100 cars accompanied by a conductor, whose duty it is to look out for accidents, the train being hard to control when the tail-rope loses its hold. The system is preferred in American collieries, and is the best plan by which branch-ways may be operated. Each branch has its own rope passing over sheaves at the ends, the principal ropes are opened at the proper points for connection with the branch ropes, the train engaged and hauled to the end of its journey.

This is an inexpensive plant to build and repair; it does not require a double track, though it demands a double length of rope. With a single roadway the sheave is vertical, and the tail-rope moves along the roof to its drum vertically above the other. In this form it is encountered in iron mines. In extensive workings it has dispensed with animal power, and is advantageously used for slight grades, and even for undulating roadways. An 18×30 cylinder, with 75 lbs. pressure, on a 2800-foot slope, grade 1 in 200, $4\frac{1}{2}$ -foot drums, $\frac{3}{4}$ main- and $\frac{1}{2}$ tail-rope, 30 trips of 17 cars each are made with a velocity of 8 to 11 ft. per second.

For passing around curves, 24-inch wheels are laid horizontally and beyond the inside rails. These carry the rope till the car reaches the turn. On reverse curves they are laid nearer the inside rail and slightly inclined to the horizontal toward the centre.

32. The endless-cable systems are much in vogue, and require less rope than the above-mentioned plan. They are very suitable for a double-track line of communication with frequent stoppages and no branches. A continuous motion in one direction is imparted to the rope by a single wheel or drum, and the tension produced by artificial means—a friction-grip or clip-wheel, or else several turns of the rope on the drum. The clip-wheel keeps the chain or rope tight, being on a car

riage frame, to the far end of which a rope is attached. This rope passes over a pulley suspending a weight, which maintains the tension desired in the tram-rope. On a single track a reversing engine works the rope alternately forward and backward, the return-rope being supported overhead or at one side of the gangway. The power is transmitted by a rope or chain suspended above, resting on top or supported below the cars;

1. In the cheapest, most universal, and effective method, the chain rests in forks riveted on top of the cars, which are singly attached at intervals of 25 to 100 ft.

2. For heavy grades, with intermediate stations the chain runs on rollers underneath the cars, a short length of chain being used for connection.

3. For a uniform grade and sharp curves the cars are attached by chain to an endless rope above them.

4. For varying grades and curves without branches, the cars are singly operated as on surface cable roads.

(1) This system supplies a continuous power which may be taken off at any point. The cars, readily connected and disconnected, are distributed singly along the line, from 20 to 100 feet apart, and as the velocity does not exceed 3 or 4 miles an hour, the boys have ample time to hitch them on the rope. The capacity is independent of the length, being determined only by the number of cars delivered. The power engines have heavy fly-wheels for regularity and compactness. A sprocket-wheel keeps the desired tension and prevents slipping. It is a driving-wheel of 3 feet or more diameter, carrying forks set radially, and capable of being screwed out and in; these are turned a little to seize the chain as it lengthens and drags, until they are paid out to the limit, when a few links are removed and the forks adjusted. Extensions are easily made when required; this is not possible with tail-rope.

This system costs less for power than tail-rope, and admits of sharper curves and steeper grades, but requires two lines of rails.

As an example of a plant: a 5773-foot road with 70° curves, cars (0.5 ton each) 50 feet apart, at 2.8 miles per hour, delivers 1250 tons per shift. The

chain shows a tension of 4000 lbs. The engine is 17 horse-power, has a drum of 5 feet diameter, lagged with wood, lasting a year (costs \$25); the sprocket (\$80) wears out in four months.

A "tail-rope committee" of the North of England Institute of Engineers reported (vol. xvii. of its Transactions) that, "as far as the cost of maintenance and working expenses are concerned, this endless-chain system can be applied, with few exceptions, to every condition of wagon-way with greater economy than any of the other systems." It is not restricted by grade, nor by any irregularities or crookedness in the roadways.

(2) The friction, and of course the wear, is much greater than in (1).

(3) The mechanism of the endless-rope system differs little from that of (1) except in the method of connection. The cars at intervals are hooked to the short chains pendent from the rope, or a small chain is wrapped twice around the rope or into loops along it. The speed is lower and the cost is higher than in (1). The first cost of one of 900 tons capacity, 3200 ft. long, is \$4000, and its yearly repairs amount to \$200.

(4) Each car is connected by a hand-clamp, somewhat like that in use on the surface cable-roads, similar in action to a pipe-tongs, and is also provided with a device to keep the rope on the rollers. The clamp is detachable. At each station or branch a man knocks off the grip and switches the cars. Instead, often a train of 20 to 50 cars is hauled from a guide-car, which is under control of the gripman who rides it.

Fuel, labor, etc., of a 450-ton endless ropeway is about \$7.50 per day. The initial cost is about \$6700 for a plant of 4200 ft. double track (with a grade of 40 feet in 1500, the rest level), a $\frac{3}{4}$ rope at 260 feet per minute, and a 12 $\times$ 20 cylinder at 56 revolutions, geared to 6-ft. drums at 14 revolutions.

For cable-roads the ropes run on grooved rollers, 6 inches diameter, 30 feet apart, resting on spindles supported between longitudinal oak stringers across the ties, as nearly central as possible. Around curves the wheels are 12 inches diameter, bolted flat to each tie, and have the upper rim smaller than the lower one, to let the grip pass easily.

The safest mode of connecting ropes is, of course, by splicing, but sockets are convenient where shortening may be required.

An electric system along the tramways is requisite for safety as well as for signalling, though the malicious destruction of insulation, etc., has caused its abandonment by many operators.

EXAMPLE.—A one-mile endless rope, travelling at 2 miles an hour over an average grade of 3 in 100, delivers 50 tons per hour. Required, the total resistance on the line and the size of the cylinders under 50 lbs. boiler-pressure and 160 ft. piston-speed. Each car weighs 800 lbs., and carries 2000 lbs. Along the line are distributed 25 loaded and 25 empty cars, about 100 feet apart, and a ton is delivered every 36 seconds; with a coefficient of friction of 0.02, the frictional resistances on the halves of the line are 1400 and 400 lbs. respectively. To raise 25 tons up the plane requires a force of 3 per cent of 50,000, or 1500 lbs.; the gravity component of the cars is 600 lbs. The gravity components of the cars and the rope, up and down, balance each other, leaving the work of the engine to be that of overcoming $1500 + 1800$ (the drag of the rope is assumed at 3000); the which, carried at a rate of 176 feet per minute, requires 1,109,800 ft.-lbs., or 33.6 indicated horse-power. $6.283 \sqrt{fk^2psn}$ is the dynamometric power doing by the steam. Substituting, k is about $7\frac{1}{2}$ inches, and for 130 strokes, s nearly 15 inches. If the driving-sheave is 6 feet diameter, it makes 9.3 revolutions per minute, and is therefore geared 1 to 7.

EX. 13.—A haulage-engine having cylinders 24×48 inches, at the head of a plane of 10 per cent grade, is directly connected to its drum. With an effective steam-pressure of 45 lbs. per sq. in., a piston-speed of 350 feet per minute, and a train-speed of 8 miles an hour, what is the size of the drum and the capacity of the plant?

The total piston-pressure is 27,150 lbs., assuming a modulus of $\frac{3}{8}$, which, at 350 feet per minute and a train-speed of 704 feet per minute, represents a continuous load of 13,497 lbs. This tension upon the hauling-rope, due to the component, parallel to the plane, of the weight of the rope, cars, and load, plus their friction, is therefore limited to 13,497 lbs. For this working load the rope may be $1\frac{1}{8}$ inches diameter, weighing 2 lbs. per running foot; the cars may be assumed to weigh one half that of their contents, W , and the frictional resistances may be allowed for at the rate of 50 lbs. per ton of normal pressure. Then, by formulæ on pp. 97 and 142,

$$13497 = \frac{100}{1000} (12000 + \frac{3}{8} W) (1 + 0.025).$$

W becomes 79,840 lbs., and the hourly capacity less than 160 tons, without allowing for delays.

EX. 14.—To ascertain if the engine is proportioned to starting the given load, we examine the table on page 97, and note that the coefficient is 0.3974.

which, multiplied by $45 \times (24)^2 \times 4$, gives a minimum moment of 41,202 ft.-lbs. With a direct-acting engine the drum would require to be $5\frac{1}{2}$ ft. diameter. The engine can therefore start from the bottom a load, W , greater than 99,300 lbs.

Ex. 15.—Should the plane be double-tracked and the engine be supplied with two drums fast on the same shaft, the only work falling upon the engine will be that due to the contents of the cars plus the total friction on the double line. In this case 7 full-length trips might be made per hour, thus increasing the hourly capacity to 377 tons.

Ex. 16.—Required the size of a haulage-plant for a tail-rope system to deliver 1000 tons in 10 hours over a road the first 600 feet of which has a dip of 4 feet per 100 with the load, the next 2100 feet a grade of 3 ft. per 100 against the load, and the lower 600 feet 2 ft. per 100 against the load.

Assume that each car weighs 1000 lbs., and that its capacity is 2000 lbs.; assume also a piston-speed of 200 ft. per minute and an average train-speed of 6 miles an hour.

On the lowest section, gravity produces a tension on the main rope of 5665 lbs., and the engine has to perform 2,991,120 ft.-lbs.: on the next upper section for 4 minutes 122.6 h. p. is necessary to pull the train up grade. At the head of the gangway the grade favors the loaded cars, which produce a tension on the tail-rope of over 6000 lbs. With a drum of 132 inches diameter, making 15.3 revolutions per minute, gearing to a pinion of 0.40 in diameter, a modulus of $\frac{3}{8}$, and an effective steam-pressure of 40 lbs., the diameter of each cylinder would be 21 inches and the stroke 30 inches.

Penna. Mine Insp.: Endless Rope Haulage, D. H. Thomas, 1891, 492; Standard Cars, John M. Watt, 1884, 214a; Self-oiding Wheels, G. M. Williams, 1889, 110.

L. Sup. Min. Inst.: Electric Haulage at Ishpeming, Mich., IV. 9; Electric Haulage at Lake Angeline, IV. 21; Elec. Loco., IV. 12 and 20.

Ill. Mining Inst.: Elec. Loco., I, 347; Haulage at Kansas Coal Co., III. 2, 117.

Trans. M. & M. Eng.: Suspended Smooth-rope Mine Haulage, XLV. part 5, 121.

Coll. Eng.: Haulage, Compressed Air, J. H. Bowden, May 1896, 228; Electric Haulage, Allegheny Co., Pa., Moon Run, F. C. Whitmore, Feb. 1897, 312 and 281; Electric Haulage, Corona, Ala., Wm. M. Brewer, Feb. 1897, 314; Sheaves, Pulleys, Track, etc., at Leith Coal-mine, Pa., H. L. Auchmuty, Aug. 1896, 6; Self-acting Switch, Baird Halberstadt, Nov. 1895, 78, J. J. Ormsbee, 80; Standard Cars in Anth. Mines, New Castle, Colo., R. M. Hosea, May 1897, 427; Mine Mules, J. E. Scott, July 1896, 277.

Amer. Mfr.: Improvements in Coal Haulage, Reuben Street, Jan. 1897, 48.

Franklin Inst. Jour.: Return Circuits on Electric Railways, Charles Hewitt, July 1896, 51.

Coll. Guard.: Electric Traction, Joseph Libert, Dec. 1894, 1154; Crossing Ropes on Haulage Planes, M. Combalot, Jan. 1897, 131; Tipping Cars, James Rigg, April 1897, 823; Horses, haulage, M. Boissier, LXXII., Oct. 1896, 734; Selection and Care of Horses, M. E. Boissier, June 1897, 1173; Haulage Tests, Dynamo—Truck-measuring resistance, M. Damon, Nov. 1896, 981; Electric Haulage, Stationary Motor, Correspondent, May 1897, 855; Haulage by Electric Transmission, details, sheaves, Herr M. Dickmann, Dec. 1896, 1154; Hopper Car, Detail Drawing, Charles Hunt, April 30, 1897, 809; Dumping, James Rigg, April 1897, 824; Electric Haulage, Sydney F. Walker, June 1897, 1052; Haulage at Cannock Mine, R. S. Williamson, Nov. 1896, 1016; Enumerations of Electric Haulage, Herr M. Dickmann, Dec. 1896, 1154; Tail-rope System, description of plant, R. S. Williamson, Nov. 1896, 1016; "Elswick" Wire-rope Haulage Clip, A. Finney, 1897, 34.

Coll. Man.: Accidents in Wire-rope Haulage, Henry Bradford, Dec. 1896, 613.

School of Mines Quarterly: Electricity for Underground Haulage, Rich. A. Parker, XVI, 37.

E. & M. Jour.: Electric Haulage Plant, Berwind-White Collieries, T. W. Sprague, LIX. 508; Large Electric Locomotive, LVI. 59 and 476; Electric Locomotives for Mines, LIX. 33; Compressed-air Locomotive for Mines, LX. 127.

Fed. Inst. M. E.: Endless Rope Haulage at Thorncliffe, Rockingham and Tankersley Collieries, W. Hoole Chambers, III.; An Underground System of Haulage, John Nevin, III.; Electric Haulage at the Cannock and Rugeley Collieries, R. S. Williamson, III.

Amer. Inst. M. E.: Wire-rope Haulage and its Application to Mining, Frank C. Roberts, XVI. 213; Electricity and Haulage, Francis A. Pocock, XVIII. 412; Note on the Friction of Mine-car Wheels, R. Van A. Norris, XVIII. 508; Electric Locomotives in German Mines, Karl Eilers, XX. 356.

CHAPTER IX.

SURFACE TRANSPORTATION.

33. The pioneer burro; aerial tramways; description of the Bleichert, Hallidie, and Huson types; capacity, cost, etc.; regulation of the tension of the rope. 34. Wire-rope transmission of power; pulleys, sheaves, rope, etc.; formulæ.

33. MINING in the mountainous regions encounters difficulties in the transportation of the product and supplies, which are not readily overcome. Often a mine is inaccessible to wagons, and burros constitute the only means of transportation. The ore is carefully sorted, sewed up into sacks containing 90 lbs. each, and packed, one on each side of the jack, to market. They travel in trains of 20 to each driver, averaging about a mile an hour, and return at the same pace with the supplies for the mine. The cost of filling and sewing the sacks and their repair is high; and as it takes 11 jacks to "pack" a ton away and fetch enough fuel to run a 26 horse-power engine 24 hours, it may readily be understood why the much-abused, patient brutes remain only in isolated camps as companions to the pioneer prospector, for whom they continue to do service between mine and wagon or mill.

For larger output they are replaced by an aerial tramway, which is a sort of an endless rope-way that can be run night or day in all seasons, without road or expensive machinery, and furnish a cheap, convenient conveyance for ore and supplies, down a declivity, around bluffs, over intervening hills, and around flat curves for a mile or more. When the grade to the point of delivery is about 14 in 100, the tramway is self-acting, the speed being regulated by a brake; below this, auxiliary power is applied to the rope at the upper end.

There are two varieties, one represented by the Bleichert, and the other by the Hallidie and the Huson patents. In the former, one or two ropes are stretched tightly and supported by standards, 10 or more feet high, to give a continuous slope from the mine to the discharging point, where they are well anchored by screw-rods and buckles. The cables constitute the roadway for the trolleys, from which the tubs are suspended. The trolleys are operated by a single or an endless rope which passes around clutch-sheaves at the top and bottom. The tubs carry as much as a ton, and dump automatically into a bin or wagon. The cost of this system is quite high, but it can handle 1000 tons per day. The carrying cable (Fig. 71)

TRAMWAY CAR FOR THE TRANSPORTATION OF COAL, ORES, SANDS, &c.

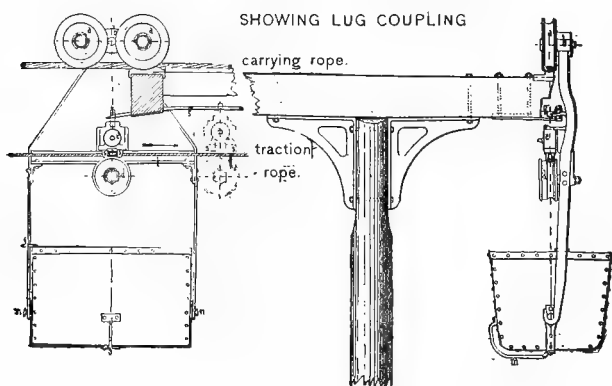


FIG. 71.

is stationary and about $1\frac{1}{2}$ inch diameter, though it is locally strengthened according to the strain to be carried. A line of steel rods may replace it for short spans and light loads, but cable is better, as tending to convert the otherwise transverse strains into tension.

In a somewhat different style of aerial tramway, one or two ropes stretch the entire length to constitute the guides for one or two large skips holding a ton or so, and attached at the end

of a rope. They are operated like the gravity planes, p. 165, and may be self-acting or not. See Fig. 72.

In the Hallidie or Huson designs, a single endless wire rope, Fig. 73, is supported at intervals of 150 to 300 feet, on suitable sheaves, which are mounted vertically on the ends of cross-arms fixed to the necessary posts or frames, and

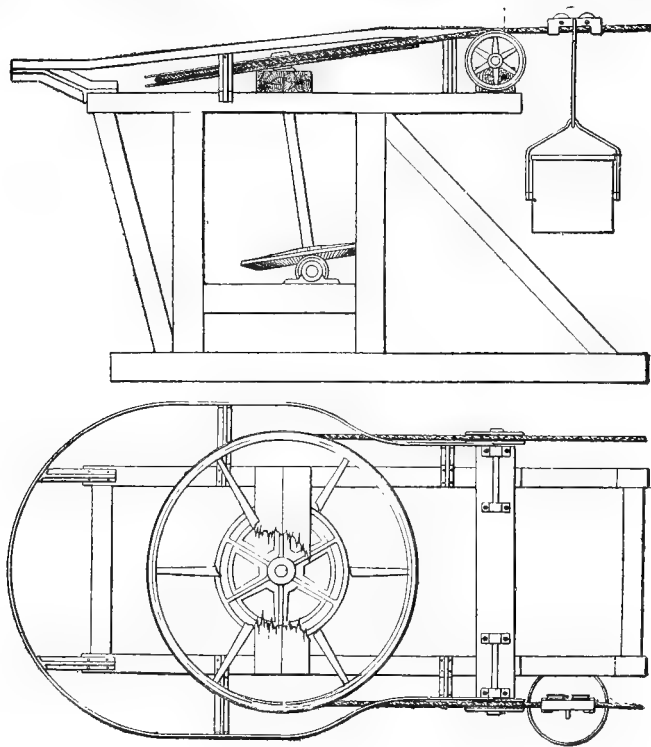


FIG. 72.

at sufficient height to clear all surface obstructions. At both ends of the line the rope passes around clip-pulleys set horizontally. The upper wheel is placed on a frame below the level of the tunnel or shaft mouth (Fig. 72). At the lower end may be a plain or a grip wheel on a carriage tower frame, which assumes a position such that a constant tension may be main-

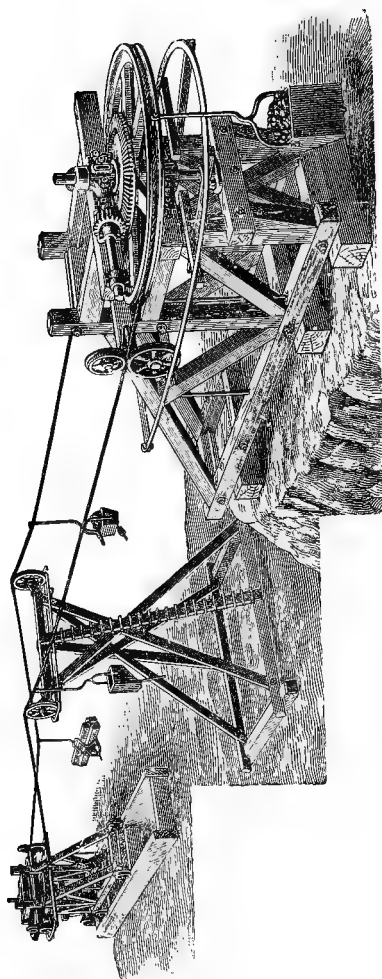


FIG. 73.

tained in the rope. Precaution should be taken to provide a hold-down rail on top of the wheels to prevent the carriage from tipping. The distinguishing feature is that the load is at once supported and moved by the same rope, which has a continuous motion in one direction, at a velocity of about 200 feet per minute. With a velocity greater than this on a steep grade, the loaded rope frequently flies off the sheaves.

Buckets of various designs, according to the character of the material to be handled, are suspended by hangers or clips, which are either inserted into the rope or clinched around the outside of it, and attached at intervals determined by the amount of material to be delivered. Usually they are wrought-iron rectangular buckets holding about 100 lbs. each (Fig. 74).

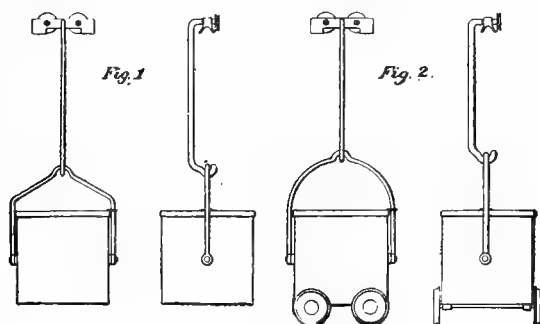


FIG. 74.

For the transport of very large outputs the buckets may be nearer together than the average 200 feet, or larger, and the rope may be heavier than the ordinary size of $\frac{5}{8}$. The buckets may be loaded at any point along the line, automatically or by hand, and are unloaded at the lower end by some automatic device. The carrier strikes a lever, which opens a catch (holding the bottom in place), and discharges the ore; a counterpoise on the bottom closes it again. The hangers are so made that they may pass uninterruptedly over the rims of the supporting sheaves and around the terminal pulleys, their consumption is large, and amounts to \$100 per year on a medium line.

The strongest form of intermediate supports are stout

rectangular frame standards of four sills, from each end of which is built an X transversely, from on top of which are heavily bolted cross-arms. As these X's lean towards each other at the top, they are not liable to get out of line, nor does the weighted side of the rope pull the cross-arm out of level. To the ends of the cross-arms are boxed the carrying-sheaves, rubber-lined and loose on the axle. To round a curve, the standards are nearer together, and the rope is slightly deflected with each wheel.

A rope-way running 200 feet per minute, carrying 100 lbs. per bucket every 100 feet, will deliver 60 tons per shift. With a descent sufficient for gravity to supply the power, three men can manage all of its operations. It requires some supervision, and delivers ore at 20 to 35 cents per ton-mile (inclusive of all allowances), and about 60 cents per cord-mile for wood. The line can be completed for \$1.30 per foot, and \$2000 for the machinery at the terminals. Curves and long stretches increase the cost; grade does not.

34. For power transference from, instead of ore transportation to, remote points, a similar arrangement is widely applied, efficient and cheap. For moderate distances, up to a mile, its efficiency is greater than by any other system; at half a mile it is 90 per cent. The inevitable concomitants—which accumulate so rapidly that for distances of over a mile electricity gives much better results—are the losses of energy due to friction of bearings, air-resistances from centrifugal action, stiffness of ropes, and elasticity, due to the spiral winding.

The rope is of the seven-wire pattern, of from $\frac{3}{8}$ to $\frac{7}{8}$ diameter, passing around sheaves, and supported over the intervening spaces by wheels. The size of the rope increases with the tension, and that, in turn, depends on the sag allowed, which fixes the distance between the stations (60 to 300 feet). The rope runs in cushioned-grooves on leather or rubber without slip, noise, or swaying, if the wheels are well-balanced and carefully aligned. Those on the driving-side are nearly of the same size as the sheaves, those on the slack-side one half smaller, the tension being less there. Where it can be arranged, the upper

side should be the slack side, and the lower, the pulling side. Large wheels are advised also, because they keep the two ropes apart.

Evidently, the power that can be transmitted depends upon the adhesion of the rope to its driving and driven sheaves. Grip-pulleys, or clutches, increase this adhesion, and through it the velocity limit. The product of the velocity and the force at the sheave-rim measures the work done. The force available at the sheave is the assumed maximum tension, T , less the loss due to centrifugal force. Not all of it can be used, because some of it is absorbed in giving adhesion, and this is an uncertain quantity.

Let v = velocity in feet per second;

d = diameter of the rope in inches;

w = weight on the journal;

W = weight of rope between stations;

j = loss, in ft.-lbs., due to journal-friction,

F = " " " " " centrifugal force;

N = the horse-power transmitted;

n = number of revolutions of the wheel per minute.

Then $F = 1.4d^2v^3$;

j = for each end-sheave, $249.V + 0.185wv$;

j = for intermediate stations, $0.1856(w + W')v$;

$N = 1.7d^3n$, when the diameter of the wheel is 165 times that of d ;

and $N = 3d^3n$, when it exceeds 200 d .

These latter are approximate values. Six- or seven-foot wheels, with $\frac{1}{2}$ " rope, at 80 to 140 revolutions per minute, will transmit 10.7 to 29.6 horse-power, while ten- to eleven-foot wheels, with $\frac{3}{8}$ to $\frac{1}{2}$ " rope, give 58 to 135 horse-power.

Tension is adjusted and maintained, as in aerial trams, by tightening-sheaves on carriages; for the rope cannot be so nicely spliced as to get the proper sag, which for spans of 150 to 250 feet should be from 1.3 feet to 3.6 feet when the rope is at rest. Every two or three months the stretch of the rope is taken up by shortening and resplicing. With inclined lines the proper deflections cannot be obtained without tighteners.

Often, instead of a continuous line and an endless rope, a series of closed ropes and double pulleys in sections do fair service, are easily repaired or renewed, and less influenced by changes of temperature.

The following list of memoirs will give aid to a further investigation of the subject matter of the chapter:

The Eng. Soc. of W. Pa.: Gravity Plane, XII., No. 9, 235.

Trans. M. & M. Eng.: Engine Planes at Wearmouth Colliery, W. R. Bell, XLV. 219.

Min. & Scien. Press: Aerial Tramways, C. T. Finlayson, at Sandor, B. C., June 1897, 544.

Coll. Guard.: Various Types of Aerial Ropeways, W. Carrington, Mar. 1897, 556.

Amer. Inst. M. E.: Aerial Wire Ropeways, J. Pohlig, XIX. 760.

E. & M. Jour.: Wire-rope Tramway, English Mt. Mine, Cal., LIX. 55, Vulcan Ropeway, San Andreas, Mexico, LVI. 615; Bleichert Wire Tramways, LVI. 394; Wire-rope Tramways, Prof. Thiery, LVI. 366; Brewer's Aerial Tramway, LXI. 230; Wire-rope Tramways, LXI. 208; Wire Tramway in the Alps, LVII. 124.

CHAPTER X.

PUMPING.

35. Exclusion of water by cribbing and tubbing shafts; building dams and plastering cross-courses in levels; the use of advance bore-holes in approaching abandoned workings; drainage by tunnels; co-operative drainage; hydraulic rams and the Hungarian system of pumping; bailing by self-filling buckets, skips, and tanks. 36. Single-acting lift-pumps; details of sizes, of rods, pipes, valves, gaskets, etc.; spiral weld *vs.* riveted pipes; formulæ for calculating the dimensions of parts; cost of surface plant; descriptions of the Cook, Wormer, and Bull pumps; working by steam or water pressure; formulæ. 37. Single-acting force-pumps; method of altering lift- to force-pump; description of the mechanism and operation of the Cornish pump; size of pipe, length of lifts, and dimensions of pump-rods; tapering rods, catches, V-bobs, and balance-bobs; formulæ for the thickness of pipes, discharge, etc.; account of the Ontario, Friedensville, and other mammoth plants. 38. Regulation of the speed of pumping; churning of the plunger, vibration of the rod, and its prevention. 39. Double-acting pumps, sinking pumps, Cushier system; steam-pumps; their construction and operation; formulæ for sizes of cylinders, discharge, etc. 40. Comparison with the Cornish pump; relative advantages of the steam plants; pumping-engines; compound and condensing pumps, duty and calculation of; rotary pumps; water-pressure engines; California and Nevada systems; electric pumps; the windmill for power.

35. TURNING to the subject of raising water from the mines, we must not forget that water gains its entrance by many and untraceable ways. In some workings it flows incessantly from some watery stratum, in others the seepage is intermittent. The subterranean current is easily excluded from the mine by the use of a cement lining, or an iron tubing to the shaft (see II, 63), but the seepage accumulates and must be pumped off, unless the workings possess a natural drainage or an easy effluence by adit or tunnel for the upper ground. A gutter at the side of the track, or under the tramway path, with a slope of 1 in 500, readily carries off the water, and not

uncommonly delivers it to a small wheel to drive a ventilating-fan. Generally the seepage, following the hydrodynamic law, increases with the depth of the opening, and a very liberal sump is provided for its accumulation. Often one shaft and its workings become, naturally, a sump for the entire district, and drain all the neighboring properties above its level, and this suggests a simple means of keeping one's mine dry. Otherwise, as the amount of water to be encountered is uncertain, provision must be made for the handling of a large volume, according to the history of similar properties. In some coal-mines of Pennsylvania as much as 4000 gallons of water are raised per ton of coal; in Colorado often 40 tons of water per ton of ore. The Ontario and Friedensville mines raise many times larger volumes. The magnitude of such work demands the employment of powerful machinery, and often on a plan too elaborate for the means of the average operator. In some localities the drainage of the district is accomplished by a co-operative scheme with extremely beneficial results. A long tunnel penetrating the country at a level much below the lowest point of exploration drains considerable territory, dispensing with the heavy individual plants, and extends the exploration and the productiveness of the mines. Numerous examples of tunnels ten miles or more in length may be quoted, some even carrying so much water as to become canals for transportation. Several such drainage tunnels are driven in the coal regions of Pennsylvania.

Upon cutting a wet cross-course to the vein, it is a common practise to plaster it up; or, in encountering old workings, to build a brick or stone bulkhead, arched convex towards the water (Fig. 197). To provide means for the escape of the accumulated water which might otherwise do injury, a cast-iron pipe is built into the dam near its top, and another near the bottom. Either, or both, may be plugged as required. Similarly, in approaching abandoned works, it is required by law in some States that a bore-hole be kept 30 to 50 feet in advance of the drift, and flank-holes on each side, to guard against dangers from the sudden breaking into the reservoir.

Under certain conditions, in stratified regions, a hole is drilled from the sump down to some permeable stratum, into which the water is discharged.

When the surroundings are such that a tunnel may not be used for the unwatering of the mine, pumping arrangements are indispensable. The earlier forms were crude, the engine being of recent date. Surface water-falls were employed to operate wheels, which raised bucketfuls from below; or, the surface water was arranged to compress air in a reservoir at the surface, from which pipes to the sump conveyed the compressed air, the elastic force of which, in turn, forced the water up to the surface through another pipe. This is a wasteful system and intermittent, but doubtless was cheaper than any other means then available.

At the Comstock mines a sort of hydraulic ram is used, by which 1800 gallons are pumped from the 2600-foot level to the Sutro tunnel at 1600 feet. The air-pressure in the accumulator is 960 pounds per square inch, and the pipes at the bottom sustain a pressure of 2000 pounds. The engine-pressure is 80 pounds, and the actual duty given, 35 horsepower per ton of coal. This has just been introduced at Eureka.

The efficiency of the ram diminishes with the ratio between the quantity of water raised and that used. With a fall of 1 and a lift of 4, the efficiency is 86 per cent; if the lift is ten times the fall, it is 53 per cent; at 1 to 20, it is 17 per cent; and with 1 to 26, it is 0.

Small volumes of water are handled by buckets, obtainable of any size, and with a capacity up to 200 gallons (Fig. 75). At the bottom is an inlet valve by which the tub is quickly filled as it sinks into the sump; it is then hauled up, its valve closes, and at the surface it is

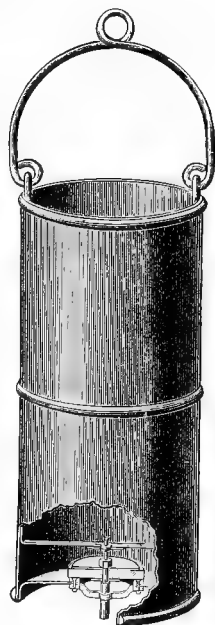


FIG. 75.

discharged by being brought down on a pin which again opens the valve. In some mines the water-bucket is attached underneath the cage, and travels continually with it. Bailing-tanks (Figs. 76 and 77) holding 450 to 900 gallons, with balanced

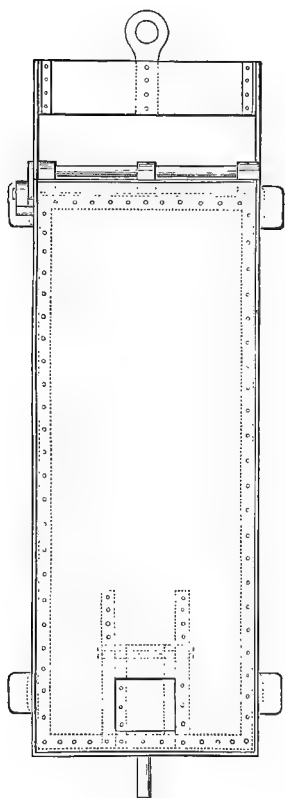


FIG. 76.

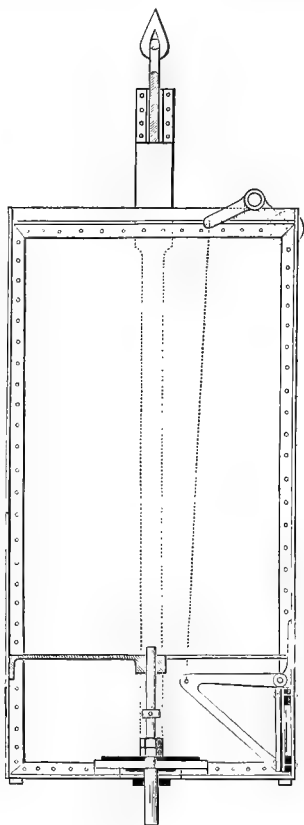


FIG. 77.

compartment of the shaft, and manipulated by an individual drum, give great satisfaction in many properties. Slopes are equipped with a similarly valved skip, the emptying being done from the mouth, as with ore. But if the mine makes more water than can be handled by these means at

spare hoisting moments, special machinery is added, and of one of two kinds: the single-acting lift-pump, or the force-pump, single- or double-acting.

36. Pumps of the first class are much in favor because of their simplicity. Their use is restricted to vertical shafts and a lift of less than 300 feet. A plunger-rod terminates in a piston in the bottom length of a pipe, where it "sucks" up from the sump water which, with the next up-stroke, is lifted into a stand-pipe, from which it is ultimately discharged at the surface.

The stand-pipe, of a diameter commonly 10 inches, often as much as 20, extends from bottom to top. It is of cast-iron, lap-weld, wrought-iron, spiral riveted seam, or weld-steel, procurable in lengths of 5 to 20 feet. The cast-iron pipe, having a smooth interior and uniform diameter throughout, is preferable and more convenient than the riveted pipe (Fig. 78) or the lap-weld iron (Fig. 79); but as it represents too much dead-weight for the strength, its days of utility are nearing an end. The ideal pipe is of steel, which gives the lightest, strongest, and most durable tubing; this may be had in four grades, light to extra heavy. It is made of spirally-laid sheet steel riveted at the overlapping-joints or cold-hammer welded. The pipes are united by bolting together at the flanges, which are riveted, screwed, or locked on the pipe (Fig. 79); or, preferably, they are coupled on the hub-and-spigot plan of sleeve (Fig. 80). This is a double socket, into which the pipe is slipped, "oakumed," and leaded from each side, as shown. For this joint the pipes have expanded ends.

A water-tight joint is secured by placing rubber, leather, lead, or, best of all, corrugated copper gaskets between the

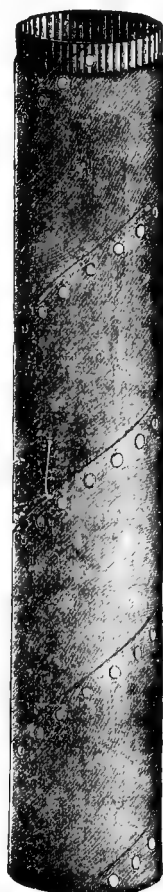


FIG 78

flanges, which are then bolted together while lowering. Spence's metal, used as a calker, offers an excellent joint, is cheaper than lead, and ought to be better known.

The pipes last fifteen or twenty years unless the water is

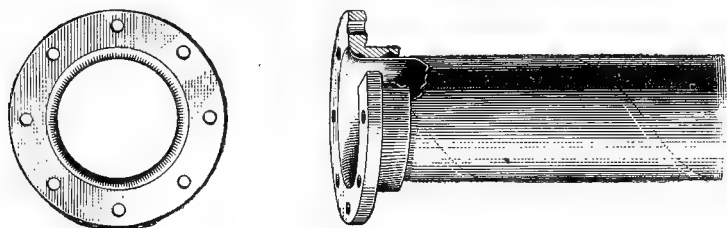


FIG. 79.

corrosive, in which case gun-metal is used. If the water is very bad, wooden pipes are made by hollowing the trees, fitting the joints, tarring them, and strengthening by wrought-iron bands at every three to six feet. In many mines recourse has been had to these as the only stand-pipe that will last over six weeks.

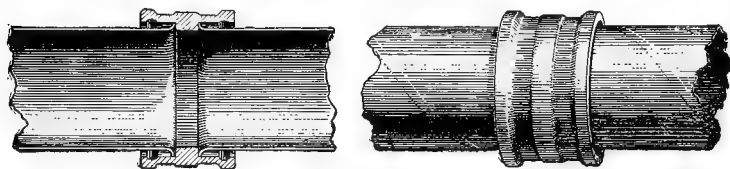


FIG. 80.

At the lower end of the stand-pipe a 12-foot length of cast-iron constitutes the working-barrel, in which oscillates a piston carrying an upward-opening valve, similar to that at the lower end of the barrel (Figs. 81 and 85). For acidulous waters the barrel is bushed with gun-metal. It should be thick, to admit of being bored out several times, as it is rapidly cut away by the gritty waters during sinking.

The valves are made of several thicknesses of oak-tanned leather cut into discs, tacked together, and slipping easily on a grid at the top of a cast-iron cellular ring-bucket. A perforated cast-iron guard on the grid limits the rising of the valve as the water passes through the bucket. These lifting-clacks are raised clear of their seats by the rising water, and open as

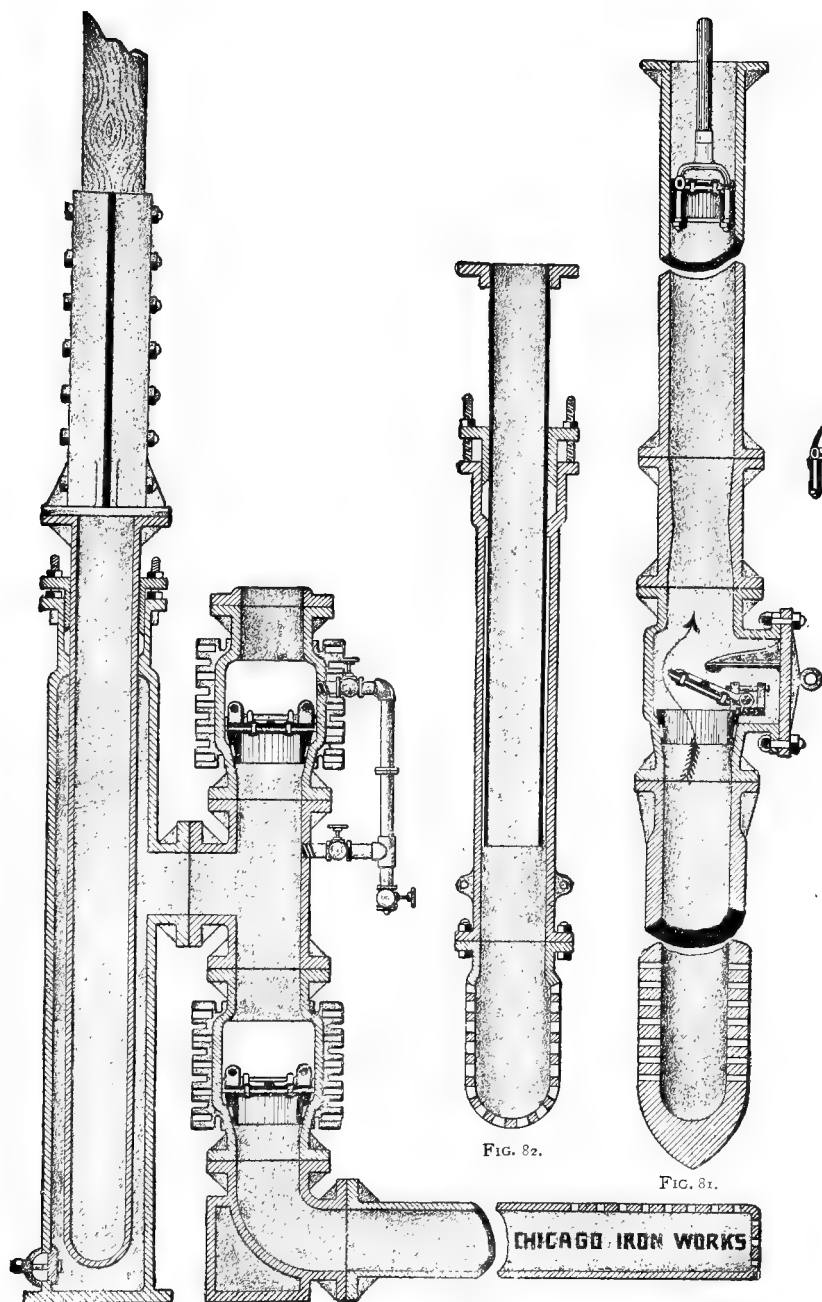


FIG. 82.

FIG. 81.

FIG. 3.

widely and shut as quickly as possible. The cellular-ring bucket casting is all there is of the piston, which fits fairly well in the barrel, and has no other packing than that offered by the leather discs forming the valve, and which are cut larger than the cylinder. The rapid movement, the wear, particularly during sinking, and the heavy pressure upon these valves, consume a set once every two weeks, or oftener. Substitutes have been suggested, amongst them flexible brass or gutta-percha plates, but they have not proven good, nor have the brass balls or conical poppets had any marked success. The valves are repaired or replaced by raising the entire pump-rod, opening the standpipe or opening a bolted door-plate in the barrel opposite the valves (Fig. 81).

Below the barrel is a length of pipe or flexible hose dipping into the sump and receiving the water through a perforated strainer. During sinking this suction-pipe must follow the lowering of the sump, and while blasting it is raised for each shot or boarded over. The flexible hose is preferable, because it can be bent and adjusted to lie on the bottom of the shaft, or hang vertically in the sump. It is of wire-wound rubber and canvas hose, which will endure considerable hard usage, and cost, for a 14-ft. length of 10 inches diameter, \$65. Without this the only way to keep up with the sinking is to use a telescopic joint on the working-barrel, allowing for say 10 feet play (Figs. 82 and 85). When the water-level has receded beyond the mouth of the suction-pipe, a length is added to the stand-pipe at the surface. The working-barrel can never be more than 28 feet from the sump-level; in mountainous districts still less; at 5600 feet altitude, 23 feet; and at 10,000 feet, 18 feet. Usually the working-barrel and suction-pipe are suspended by chains from two stulls resting in the cribbing, and the stand-pipe supported at intervals by stout reachers.

The piston, or "bucket," is attached by an iron fork (Fig. 84) to a wooden rod 4 or 5 inches square, extending up through the pipe to the surface where it is connected either with one end of a walking-beam or to the piston of a single-acting engine. As it receives a tensile strain, the joints are scarfed and strapped,

or, if the ends are flushed, two continuous lines of strap-iron breaking-joints are bolted together through the rod. The latter plan reduces the breakage and the number of stoppages for repairs. (Drill the bolt-holes in the iron; never punch them; and keep a good set of taps, dies, and drills for this work, also a good iron crab or winch.) A 4-inch rod is large enough for a 12-inch pipe; and a 5-inch, properly spliced and strapped, for a 13-inch to 16-inch delivery. The size of the straps is easily calculated. The area of each one, a , should be $d^2 D \div 40,000$. A 200-foot pump-rod requires two straps $4 \times \frac{1}{8}$ or $3 \times \frac{1}{8}$ for a 10-inch pipe.

At the surface the column-pipe terminates in an elbow discharge or in a laundry-box and trough, the pump-rod continuing up to the framing. The mechanism by which the motion is communicated to it is simple. A stout frame, with two samson posts, supports a working-beam receiving its oscillatory motion from a pitman actuated by a crank-arm, adjustable to a 1-, 2-, or 3-feet radius, giving strokes of double this length, at the opposite end, to the pump-rod, which requires little force besides its own weight. The arm is on a shaft turned from the engine by cog, geared 1 to 6 or 7, giving 12 to 20 strokes per minute to the rod. The iron-work of this frame, inclusive of cogs, pulley, and castings, will cost about \$250. The wood-work, including a 24-ft. $\times$ 15 inches square walking-beam, about \$125. Wherever cog-gearing is required for heavy work, the author insists upon a solid hub if the wheel is not too large for a single casting.

To save the cost of this cumbrous framework and the loss of power, a steam-cylinder is placed over the shaft, standing vertically, its piston being bolted to a fork on the rod. This arrangement requires no framing beyond a solid foundation for the engine, and involves the purchase merely of a steam-cylinder. The piston receives steam on both sides, though, theoretically, it need only be single-acting.



FIG. 34.

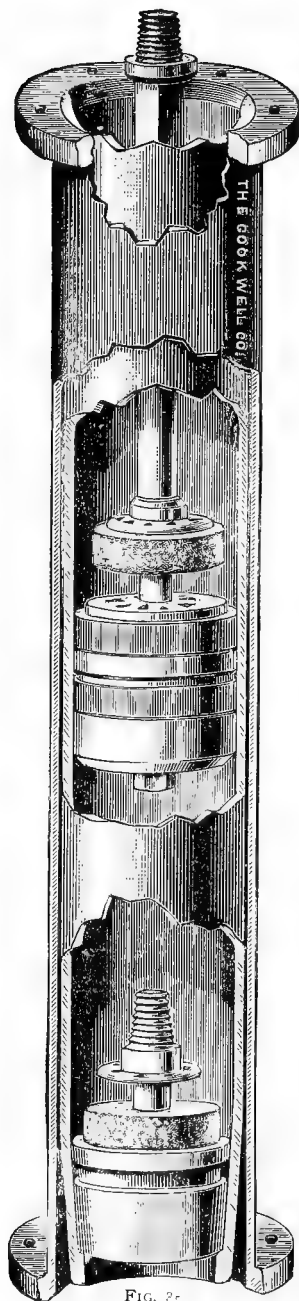


FIG. 86

It is not certain that this form gives a higher duty per bushel of coal than the drive-rod pump, for, while the friction is less, the steam consumed in the down-stroke is unnecessary, except for increasing the speed. The main objection preventing its more general adoption is the large portion of the shaft-mouth it covers. Besides, to lengthen or repair the rod or column-pipe, the cylinder must be displaced, or the additions are made below; either is slow. This pump cannot be used in slopes; the irregular wear of the cylinder on one side cannot be compensated for, nor the friction of the rod in the pipe counteracted. These cylinders are easily set, not very expensive, and work to a charm. A 12×36 cylinder, with fittings, cost \$325; larger ones may also be had at moderate prices. They are known as the "Cook" (Fig. 86), or the "Wormer" pump, from the name of the manufacturers. By the name of "Bull" pump, first introduced by Wm. Bull in 1798, they are better known in collieries, where their size is greater than, and their piston-speed about the same as, the former varieties, which run best at 24 double strokes of 3 feet each, while the latter makes 6 or 8 of 10 feet each, in a cylinder as large even as 55 inches.

Where water under considerable head is obtainable at the surface, its property of incompressibility may be utilized by admitting it under the piston to raise it, after which it flows

out. Unless kept well under control, it causes shocks and blows. A pressure of 57 pounds was obtained in a $50 \times 120''$ cylinder from a head of 132 feet, and 5000 gallons raised per minute 132 feet, by a 42'' plunger.

On the down stroke the rod falls through the column of

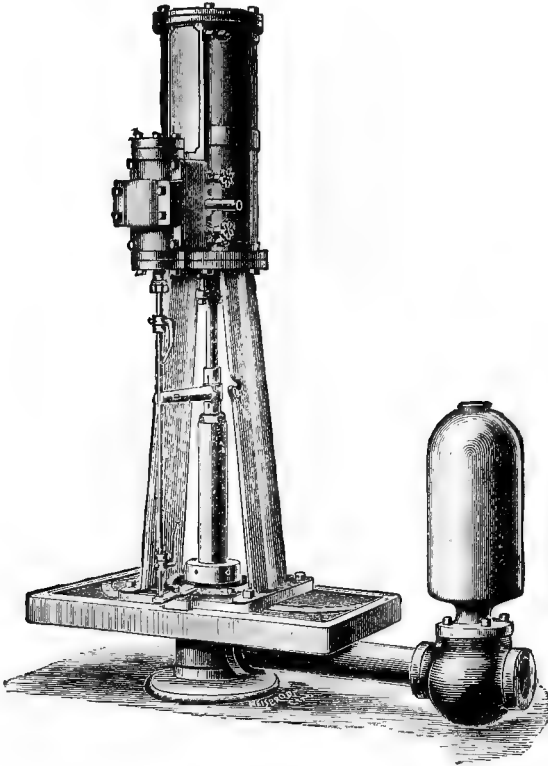


FIG. 86.

water, while the valve in its piston opens and the clack of the working barrel closes. Returning, the valve's action is reversed, water rises from the sump into the working barrel, and all that above the piston is lifted a distance equal to the stroke, and a column of water simultaneously discharged at the surface.

If d be the internal diameter of the pipe in inches, and L the stroke in feet, the discharge in gallons with each up stroke is $0.0408d^2L$; and the work

done per minute, in foot-pounds, exclusive of resistance in the cog-gear and the mechanism for transmitting the power, is $0.34272d^3LND$, where N is the number of double strokes per minute, and D the height of the water-column in feet. The direct-connection lift-pump wastes less power in friction, and has an efficiency of about 85 per cent. Its least working steam-pressure is represented by this equation: $k^2p = 0.345d^2D$.

It will be seen that with a moderate steam-pressure of, say, 80 lbs. the ratio between k and d must be large if the shaft is deep. Moreover, the size of the wooden rods for a long lift would have to be so large as to nearly, if not quite, close up the pipe. Hence, when the depth of the shaft has reached 250 feet the lift-pump is no longer practicable, and must be altered to a single discharge-force, or replaced by the more economic continuous-flow steam-pump.

36. A force-pump differs from the lift-pump in that the rod works outside, not inside, of the stand-pipe, the lower end of which is bolted to one end of a cast-iron H-chamber, of which the other stem carries a long working-barrel into which plays a solid plunger-piston (Fig. 67), instead of the bucket-valve of the lift-pump. Below the stem carrying the stand-pipe is the suction-pipe, and in it is an upward-lift clack-valve. Then during the up stroke the lower clack opens, while water rises through it into the working-barrel; as the plunger falls the water is driven through the upper valve against the column of water in the stand-pipe. As sinking progresses, suction-lengths are added at the bottom until the sump is lowered beyond the suction distance, then the pump is lowered, while additional lengths of pipe are attached between it and the H-piece at the discharge-station above, until a lift of 300 feet has been reached; then the suction-barrel is removed to do service similarly for the lower lifts and is replaced by another H-piece.

Frequently, as the mine-shaft deepens, the working-barrel and suction-length of the lift-pump are retained, the stand-pipe disconnected, and three hundred feet or less placed on a water-tight tank into which the water, lifted from below, pours through a goose-neck at the top of the barrel, and from which it is forced to the next station above. During

sinking the lengthening of the pipe between the working-barrel and tank keeps pace with the lowering of the sump until another full lift of 300 feet has been attained, when the goose-neck and all below it are replaced by the H, or tank (Fig. 89).

An H-piece and a plunger working-barrel are placed at every station, beginning a lift of 300 feet (Fig. 89). The H-piece has a door-plate opening into it opposite each valve. The valves are poppet or hinged, and on account of the heavy pressure on them—the upper one especially—they are of iron, with leather washers below. The H-piece depicted in Fig. 87 allows of the heel of the valve raising whenever chips or pebbles are caught under it. The working-barrel is about 15' long and at the top has a stuffing-box, through which works a cast-iron piston-plunger carefully turned, and of a length greater than the stroke. Lifts are rarely over 350 or less than 150 feet in height. Their number increases the first cost, friction, and repairs, but permits of greater speed and more strokes, hence smaller pumps.

Excepting the short suction lift-pump at the bottom, all the column-pipe is in one continuous line, broken only for the introduction of H-pieces or tanks at the stations. The discharge at the surface is into a laundry-box and trough. The thickness of cast-iron pipe, in inches, must be

$0.00009Dd + 0.34$; and of wrought-iron, $0.000025Dd + 0.12$. These give, for a 10" pipe of 300 feet depth, 0".61 and 0".2, respectively; their weights being 17.3 and 6.3 pounds per foot. D is the vertical head of water in feet, and d the diameter of the pipe in fractional feet.

The pump-rod extends in one continuous line down the shaft, outside of the pipe, terminating at the bottom in the piston of the bucket-lift. At intervals are offsets, to which are bolted the rods carrying the plungers. Wings, i.e., strong timbers, clamped by iron collars, are also attached to the sides of the rod. On either side of the rod, in close proximity to it, reaching across the shaft, are pairs of stout stulls, to catch the wings in case of accident, and to serve as stays to prevent buckling of the rod. The size of the rod is only a question of computation. The rod is alternately in tension from the weight of the rod while rising, and in compression from the

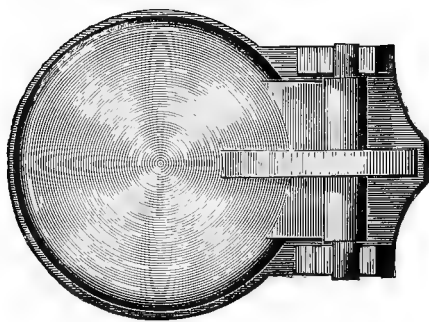
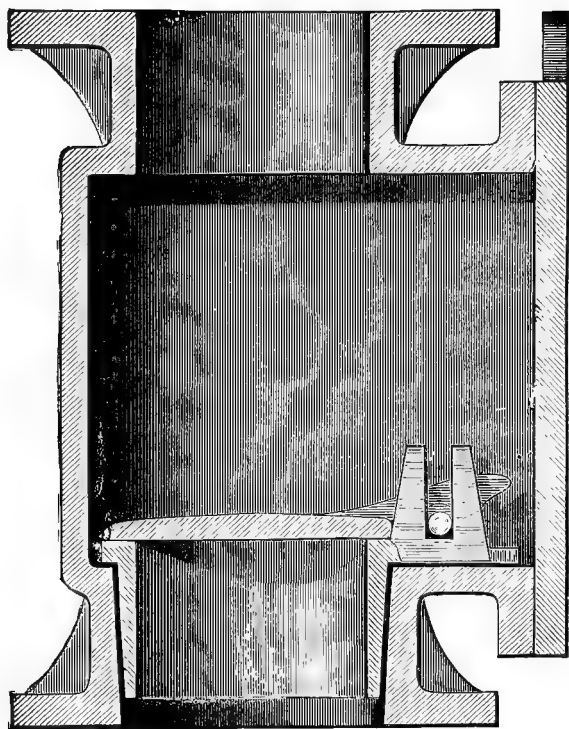


FIG. 87.

weight of the water being forced up. Its cross-section is always proportional to the weight of the number of lifts below that point plus that of its own lift. Hence it tapers to the bottom. As an instance of the size of a rod,—that of the Maira, 2300 feet deep,—we find the first 780 feet down was $16'' \times 32''$, tapering to $12'' \times 24''$; at 864 feet it was $16''$ square; it tapered to $14''$ square at 964 feet: thence it was 13 and 12 inches to the bottom. In well-ventilated shafts, wood is the preferable material for rods, neither wrought-iron rods nor wire rope having the requisite resilience.

In vertical shafts the rods fall freely by their own weight; in slopes they rest on friction-rollers, placed about thirty feet apart. When iron ropes are used instead of wooden rods, sheaves support them. Changes in the slope may be provided for by the use of a rocking-arm. A chamber is cut in the shaft at the angle in which is firmly set a frame, on which swings a V bob by a hinge-pin at the apex. To the two arms of the angle the inclined rods are attached. While this arrangement is not desirable because of the expense and the loss of power, still it is the best to be had when slopes are sunk on contorted veins.

The rod is connected at the surface to one apex of a king-post truss balance-bob. The horizontal beam is about 25 feet long, with a saddle and axle underneath near the centre. At the upper end of the king-post, which is 8 feet high, is the connecting-rod to the engine. Besides the braces on each side down to the beam, are a pair of tie-rods, taking with the braces on each side alternately tension and compression. All the members of the frame are of wood or iron, in iron shoe-castings at the ends. The frame stands vertically, in a pit dug alongside of the shaft, 8 or 10 feet deep (Figs. 20 and 88). The rocking motion is communicated to the bob by a connecting-rod, operated as a crank from a wheel geared to the fly-wheel shaft of the engine, the work of which, during the up and down strokes, is somewhat equalized by the bob and its counterpoise. The third apex of the triangle is occupied with a box full of iron and boulders, to counterbalance the excessive weight of the

rod; for it will be found that the weight of the long column-rod of a strength requisite to force the water up is much greater

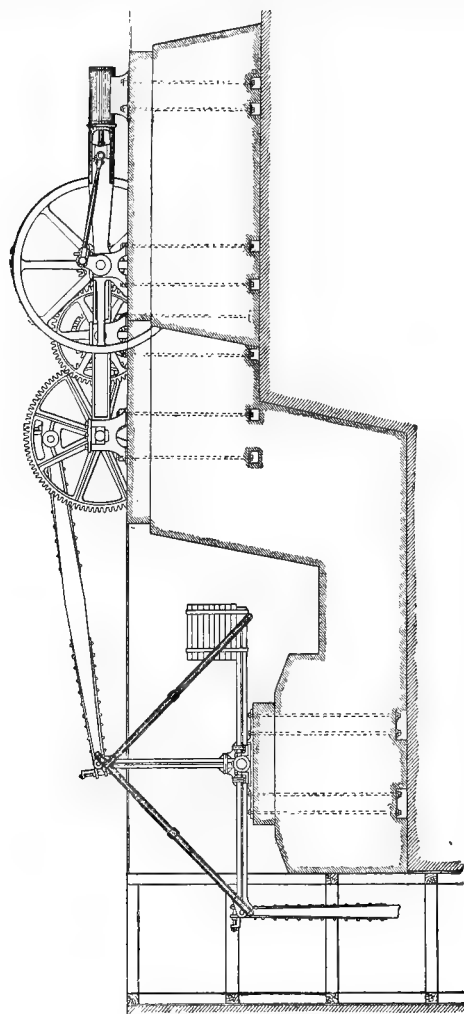


FIG. 88.

than that of the water pumped. A certain pump raising 440 gallons 1690 feet by six lifts, in a 22" pipe has a balancing weight of 33 tons on the bob.

In mines utilizing the pump-rod for a man-engine additional counterbalance-weights are connected at intervals down the shaft (Fig. 73). Sometimes two lines of rods are used in a shaft, working two pumps from the same bob, in which case no counterbalance is needed. All the foundations about the shaft should be carefully laid: condensed steam and pump-water soon makes the ground yielding. In stable ground, heavy beams buried in the stone will suffice; in ground at all soft, a secure foundation can only be obtained by concreting a considerable area, for 6 feet deep, and erecting a rigid timber- or brick-work base several feet high (Fig. 81).

38. The combination of lift and force pump as thus constructed is known as the Cornish pump. When once placed and its speed regulated, it gives little trouble. It is the most reliable and also the most expensive pump in use. It has numerous advocates as against the steam-pumps; but in transplanting the system to America we discarded its sole redeeming feature—the cataract-engine—while persistingly clinging to the worst—the cumbrous bob and rod. When the vertical direct-acting engine was introduced it was thought to be a great improvement, because of the suppression of the heavy bob; but it was soon discovered to be a mistake, and the beam was quickly re-established. An engine, boiler, and fittings complete, with three 15" plungers and one 16" lift-pipe, etc., etc., for a 600-foot shaft, weighed 650,000 lbs., had a capacity of 800 gallons, and cost in place \$54,000.

The behavior of the pump may now be easily understood. Consisting of a suction-length and a series of force-pumps, one above the other, the water is driven by stages to the surface (Fig. 89). The engine raises the rod, and water is sucked up into the working-barrel at the bottom while that above the bucket is lifting to the first station above. Here the plunger is drawing water up through the lower clack into its barrel; likewise all the other plungers. At the end of the stroke (6 to 10 feet) occurs a slight halt, incidental to the change of direction. The rod falls by reason of its own weight, and each plunger closes the lower clack in its H-piece, opens the upper one, and forces

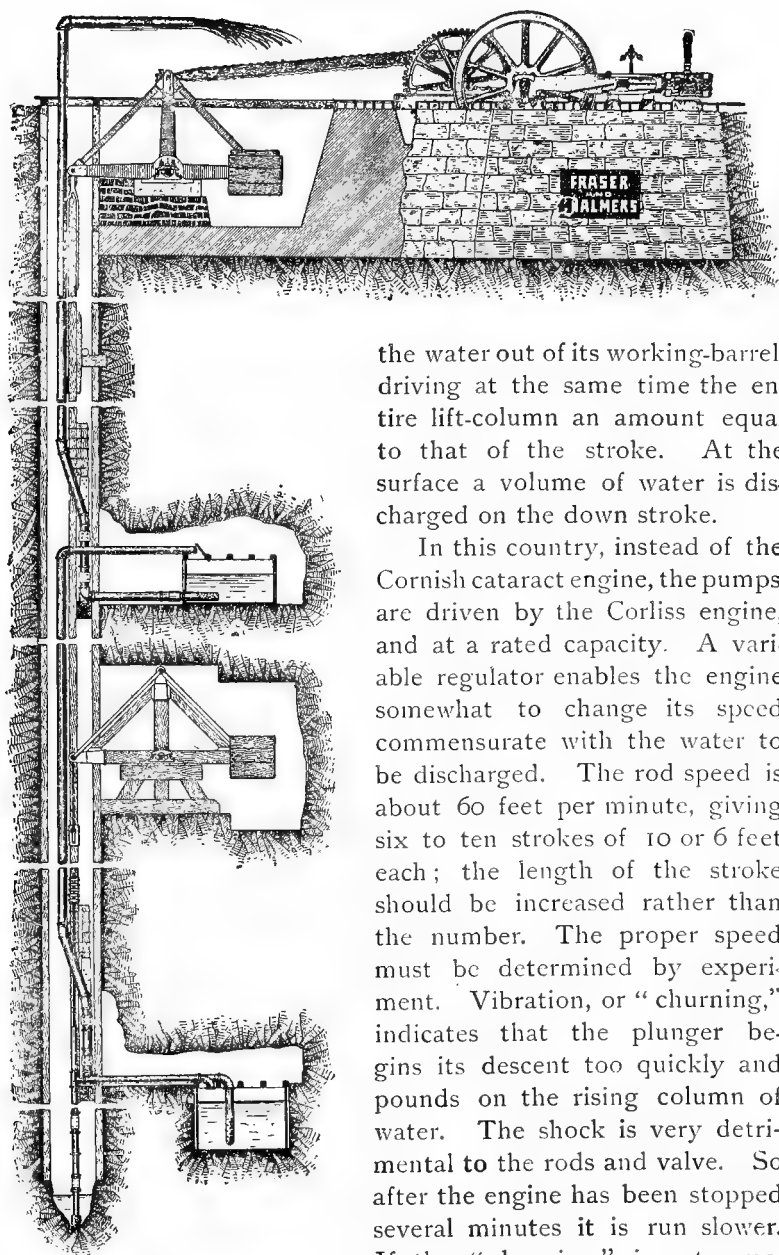


FIG. 89.

the water out of its working-barrel, driving at the same time the entire lift-column an amount equal to that of the stroke. At the surface a volume of water is discharged on the down stroke.

In this country, instead of the Cornish cataract engine, the pumps are driven by the Corliss engine, and at a rated capacity. A variable regulator enables the engine somewhat to change its speed commensurate with the water to be discharged. The rod speed is about 60 feet per minute, giving six to ten strokes of 10 or 6 feet each; the length of the stroke should be increased rather than the number. The proper speed must be determined by experiment. Vibration, or "churning," indicates that the plunger begins its descent too quickly and pounds on the rising column of water. The shock is very detrimental to the rods and valve. So after the engine has been stopped several minutes it is run slower. If the "churning" is not reme-

died, look to the valves—they may not be giving a free flow. The valves should afford unobstructed passage to the water in one direction, and close perfectly in the other,—two antagonistic conditions, which can be attained only partially. The strain on the valves is enormous, and if tried too hard they become weak, do not work properly at either stroke, and lose water by their “slip.” If the valves play all right and the vibration still continues, it may be because the joints of the rod have a backlash, or the bob requires resetting. This vibration must be particularly guarded against if the rod is also to be utilized for a man-engine. Increasing the mass of the rod is sometimes, though not always, a remedy. For convenience in repairing, the pumping compartment should be large enough for a ladder-way, with plats and chambers.

39. There have been a number of attempts to make a double-acting pump, retaining therewith the advantages of the Cornish. Its use would save space in the shaft, the pipes for a continuous discharge occupying less than one fourth the area of a single discharge-pipe and its rod.

Cushier's system of pumps for deep mines consists in having sets of two pumps, each working in concert, one above the other, the suction and discharge pipes being common to both pumps.

The pumps are placed at intervals of about 200 feet in the shaft, the power being transmitted directly through the centre of the plungers. The connection with each other and to the motive power is effected by means of a steel-wire cable, encased in wood, preventing it from external wear as well as from rust. This cable is fastened to, and its length regulated by, shackle-bolts.

The plunger of the lower pump, in a set, is double in area that of the upper one, so that in working on the upper stroke one half the water raised fills the chamber of the upper pump, the other half being forced out through the discharge-pipe on the down stroke; the upper pump-plunger forces out, in its turn, the water in the chamber, thereby causing a continuous delivery.

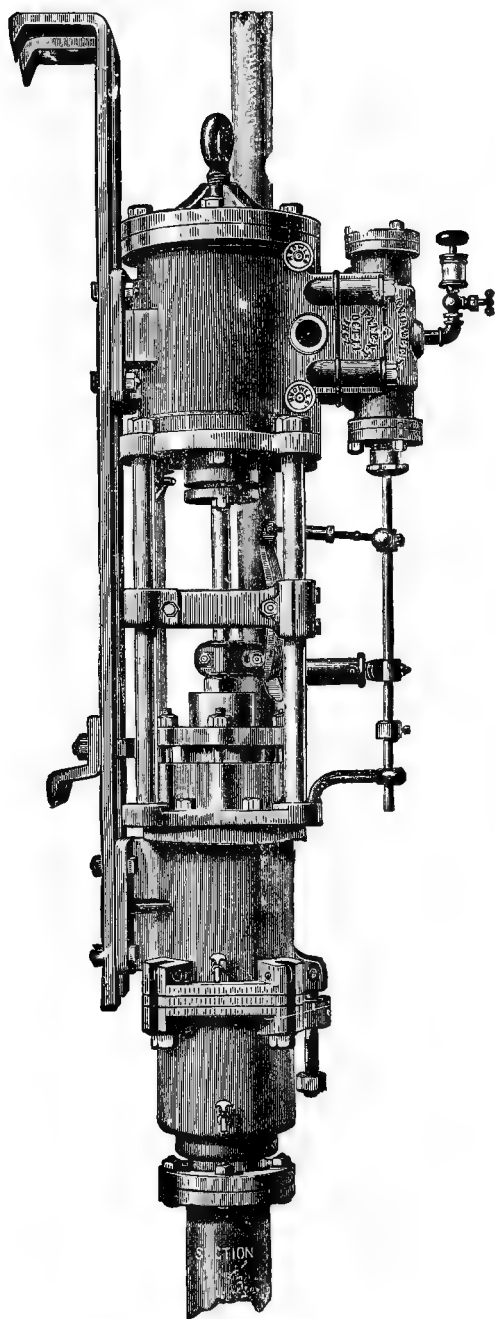


FIG. 90.

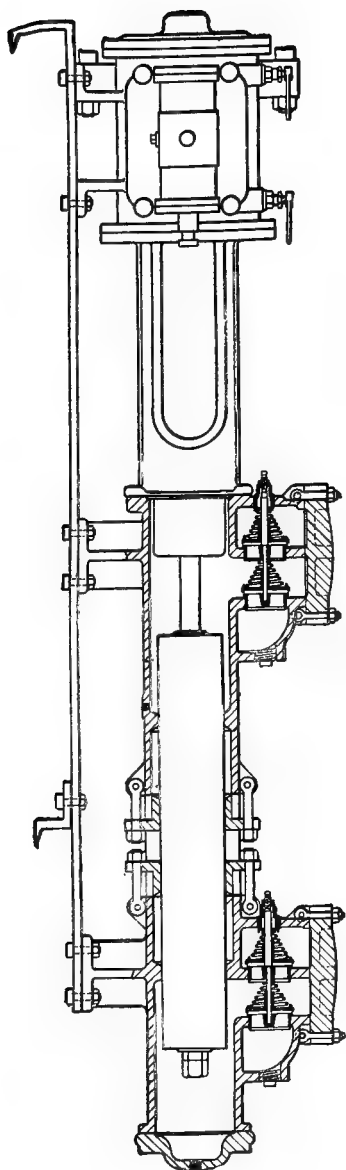


FIG. 91.

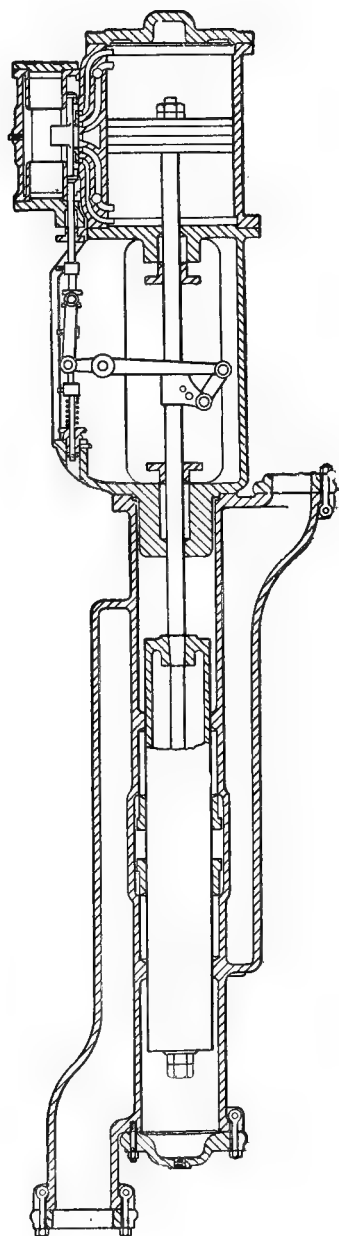


FIG. 92.

This form of pump can be worked at any angle, to any depth, and is almost perfectly balanced. The last-named advantage enables it to be connected with, and worked by, a direct-acting steam-cylinder, and thus do away with the complicated gear and bob of the Cornish.

The sinking-pump is on this plan, but not so complex for sinking or recovering mines. It is operated, hanging vertically from some support, by a chain to the bale attached to the steam-cylinder, and taking the suction in at the bottom of the pumps. By this suspension it is able to be accommodated to a varying water-level, and is used while sinking. A double-acting, centre-packed plunger is directly connected with the steam-piston, from which it receives its power, as in the horizontal pump (see Figs. 90 to 92). Cameron, Knowles, and Deane have sinking-pumps of similar pattern. The valves are absolutely positive, and are protected by a cast-iron shield serving as a yoke between the steam and water ends, while those in the steam end are cushioned to regulate the strokes. Hand-hole plates, with hinged bolts, allow of easy repair of the valves and shoes and dogs of easy handling.

The steam-pump dispenses with the cumbrous gear, bob and rod having instead a small, well-lagged steam-pipe, conveying the power down from a surface boiler. Its construction is similar to that of the air-compressor (Fig. 93), consisting of a steam-cylinder in which a piston oscillates, and by rockers moves its steam-valves without the aid of any rotary appliances; at the same time it reciprocates a solid plunger centrally in a water-cylinder, at each end of which is a set of double-beat valves of appropriate construction, open only so long as the water is being forced, and closed at once without the aid of springs. One valve is removed in the figure. In large pumps double sets of inlet and discharge, brass-covered, single-beat, rubber valves are used, closing with a spring. Where great pressure or the gritty nature of the water renders the use of the single piston undesirable, the water-cylinder is divided in the centre, and a pair of plungers, discharging alternately, work in the opposite ends, and are connected with

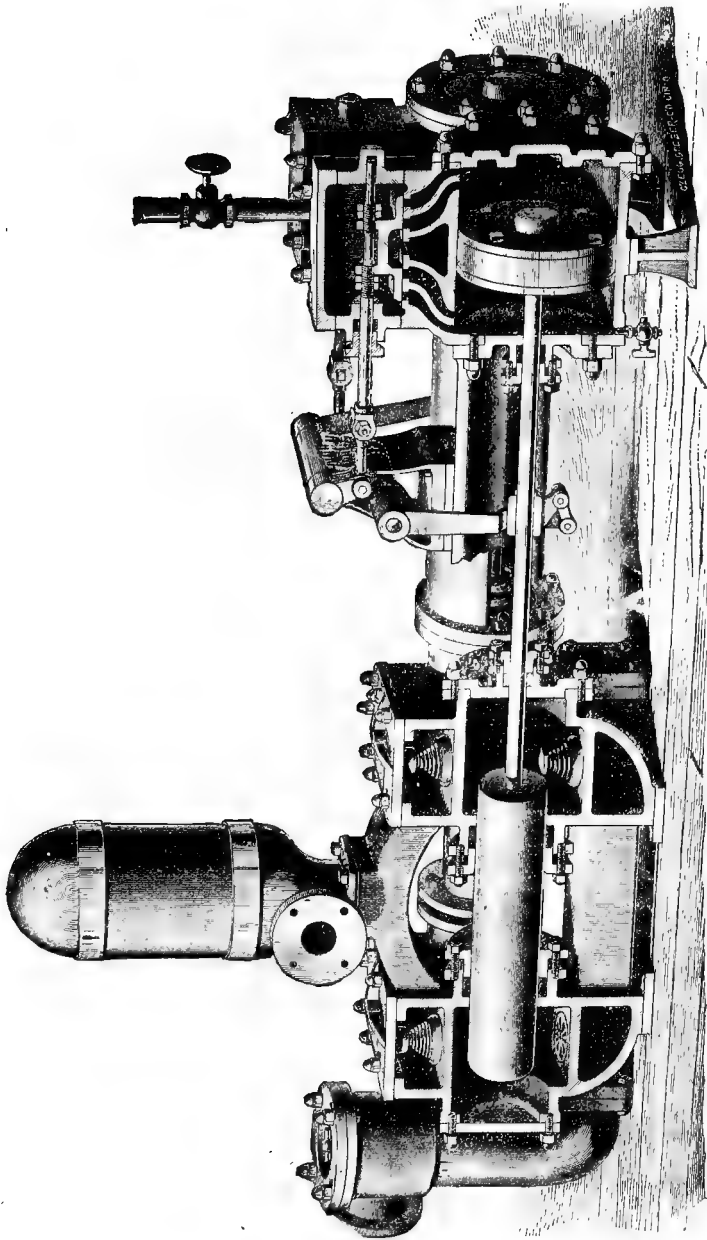


FIG. 93.

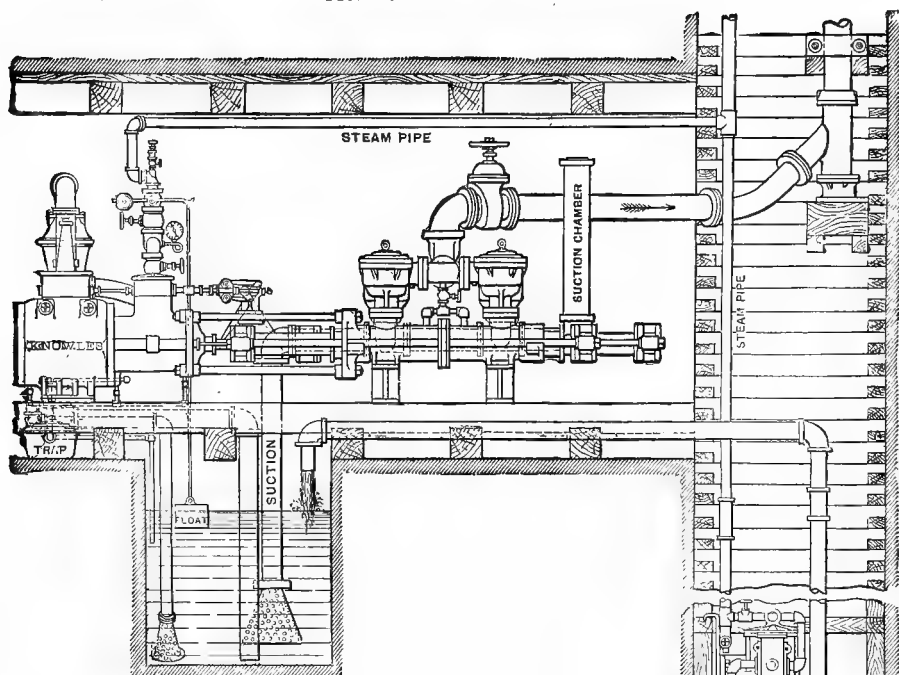
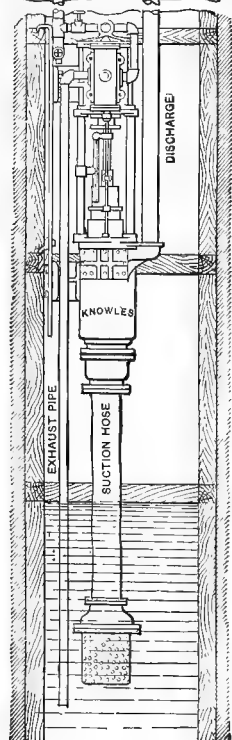


FIG. 94.

yokes and heavy outside rods to the steam piston-rod (Fig. 96). This arrangement of external stuffing-boxes permits of instant detection of leaks. Strictly speaking, the combination is a pumping-engine; but this term is customarily applied only to the double and triple expansion engines used for city supply.

The suction hose is connected under the inlet chamber, and the discharge-pipe to the surface on top or at one side of the outlet chamber. The water passages are short and very direct; the valves should be large, move quickly, and close tightly, that little loss be experienced; otherwise the effective and suction powers are both reduced. I have measured a loss on account of them as high as 7 lbs. The valves are hinged or poppet, single or double beat, of rubber or of composition metal.



These pumps operate under the direct action of the steam, the pressure of which may be within limits increased. The ratio between the areas of the steam and the water cylinders somewhat exceeds that between the height of the water-column and the steam-pressure; but when the lift becomes so great as to require a boiler-pressure of over 100 lbs., the pumps are employed in relays, each pump delivering to the tank above. Fig. 94 shows the disposition and arrangement.

The pressure on 1 sq. in. of water-cylinder of a pump = $0.434L$ pounds, where L is the vertical head. That on the bottom of a pipe d inches diameter = $0.341Ld^2$ pounds. The total pressure upon the water-piston = $0.341Lc^2$, where c is the diameter of the water-cylinder. The equilibrium of pistons of the direct-acting pump is expressed by the formula

$$fk^2p = 0.341(L + h)c^2,$$

wherein h is the lost head due to friction in the pipe, k the diameter of the steam-cylinder, and f the modulus of the pump. The number of gallons, G , discharged by the pump per minute is $0.0408c^2s$ (s being the piston-speed in feet). The indicated horse-power is measured by $0.000253GL$, where L is the height of the lift. The work performed by the water-piston = $8.33G(L + h) \div f$.

The very common ratios between the diameters of the steam and water cylinders of the pump, $k \div c$, are between 2 and 3.

Though the piston-speed may be altered as desired, the standard running rate is 100 feet per minute, and that of the water flow in the pipe 200 feet per minute. The common makes of pumps in the United States are the Knowles, Blake, Goyne, Cameron, and Worthington, differing somewhat in the form and solidity of construction.

40. For general purposes, these direct-acting force-pumps are coming into almost universal use. Their chief feature is their equal efficiency slow or fast; they are capable of quick adjustment in speed and discharge, as emergency demands; but they require close watching, especially where the water is "quick" or they may be drowned. The Cornish pump does not admit of variations in its rate: during summer and winter it is run only a few hours during the shift to empty the sump which has been filling overnight. The small cost, great simplicity, and ease of repairs give the steam-pump an important advantage. A plant with boiler, pipe, and fittings, complete,

can be installed for less than one fifth that of a Cornish outfit. One for 850 gallons per minute, 400 feet, cost \$15,000 in place.

As their maximum suction length is 28 feet at the sea-level, these pumps are placed on timber seats, in a large, well-timbered excavation alongside of the shaft and near the sump level, which must practically be invariable. They are useless during sinking without a sinking-pump to deliver to their tanks. In coal-mines, and where the machinery can be established for a permanent bed, these pumps have no rival (especially if compounded); whereas in vein-mining the pumping apparatus, and indeed all of the machinery, is continually being planned and arranged with a view to further continuation. For this reason metalliferous miners are compelled to choose between a set of relays of direct-acting pumps at each 200 or 300 feet, with a sinking-pump at the bottom, and the Cornish pump with its several force-stations and its bottom-lift.

The principal difficulty is with the disposal of the exhaust. If turned off into the sump (Fig. 94) or air-way, the temperature of the mine is raised, ventilation is injured, and the timbers ruined; if carried to the surface, the condensation in the pipe gives trouble. The best remedy is to use a condenser, which reduces the back-pressure and increases the efficiency. Jacketed compound or condensing steam ends can easily be connected to these pumps.

But whatever the fuel-saving appliances, the direct-acting steam-pump does not equal the Cornish cataract pumping-engine, though the introduction of the fly-wheel and compound cylinder gives a good approach to it. A fly-wheel is really necessary in order to secure the full benefits of a high degree of expansion, which, as has been seen on page 66, is not feasible in one cylinder, because the resistance (the weight of water forced up) is always the same throughout each stroke. A fly-wheel distributes the steam-power excess of the first part of the stroke to the latter. By the use of the compound cylinders high pressure and expansion are carried on simultaneously throughout the entire stroke, and the saving of steam-power may be fully 30 per cent. Two duplex pumps are

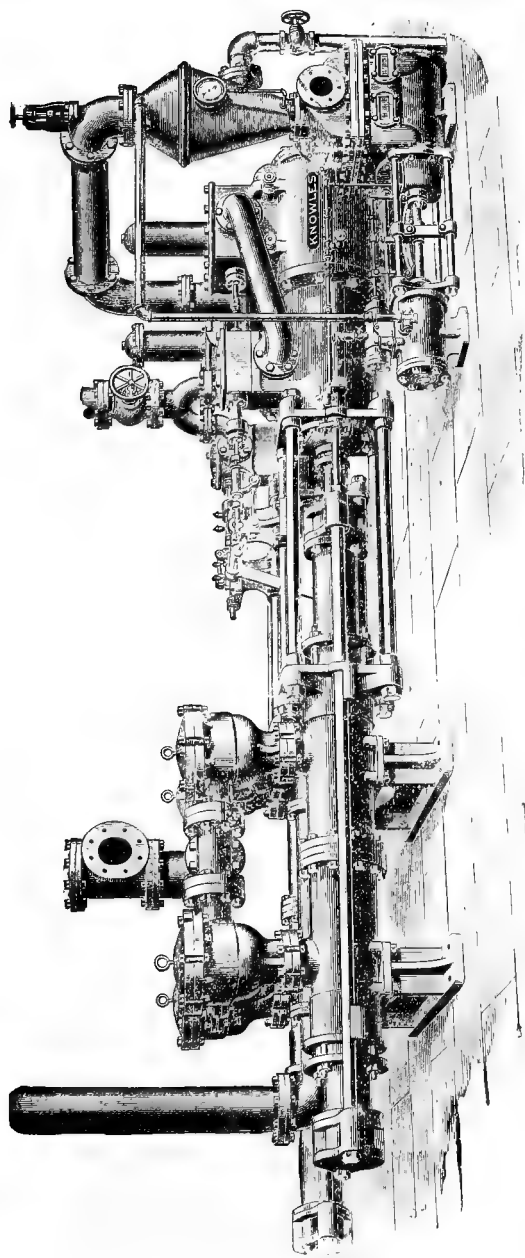


FIG. 95.

illustrated—the Dean compound pump in Fig. 96, and the Knowles in Fig. 95, both having externally packed pistons.

The calculating of the steam and fuel economy is easily made; the necessary elements are few in number, no assumptions are involved. The standard of comparison of the work of a pump is the number of million foot-pounds of work actually performed per bushel (80 lbs.) or per 100 lbs. of coal. The combustion of one pound of anthracite gives sufficient heat to, theoretically, do 12 million foot-pounds of work. The ratio

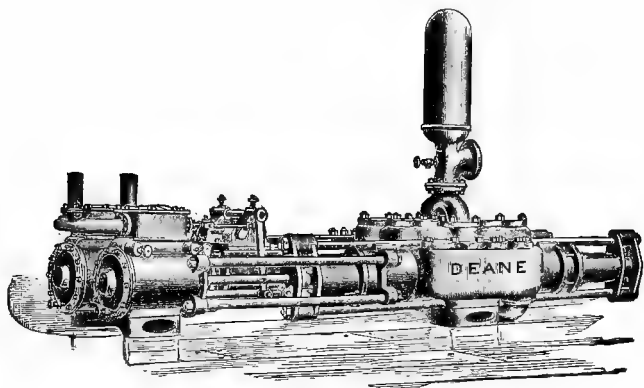


FIG. 96.

between this and the work actually done measures the efficiency or "duty," to the consideration of which in and about mines insufficient attention has been given, notwithstanding its pecuniary importance. The duty of a small pump is from 7 to 15 million foot-pounds per 100 lbs. coal; a compound gives from 15 to 30 millions; while the higher types of pumping-engines furnish from 30 to 100 million dynamic units, corresponding to the consumption per hourly horse-power of 28 to 13, 13 to 6.6, and 6.6 to 2 lbs. coal, respectively. To find the consumption of coal per hourly horse-power, divide 198 by the duty (in millions). A recent report of a Worthington engine having a capacity of over 1000 gallons per minute, against an equivalent of 2000 feet head of water, showed a duty of 184 foot-pounds per thermal unit, or 158,000,000 dynamic units per 100 lbs. of coal.

To illustrate the influence of compounding and jacketing the steam-cylinder, and of condensing the exhaust, upon the coal bills, two examples will suffice. As has been stated, many of the collieries pump 4000 gallons of water per ton of coal hoisted. To raise this only 300 feet requires the consumption, theoretically, of 336 lbs. anthracite for a daily output of 400 tons. If the duty be 90 million, as in Cornwall or by a Knowles duplex compound, or 20 million, as with our average pumps, the aggregate yearly consumption is 675 and 3005 tons, respectively, or of 900 and 4000 tons of lignite.

But duty is not the sole feature of a piece of machinery: the repairs and lubricant accounts and the durability of the plant are not to be overlooked; for the indicator-card is a less valuable guide than are the coal, oil, and packing bills. Moreover, the question of the comparative value of the inconveniences in the use of steam underground with those of the occupation of a shaft compartment by rods, catches, etc., has some monetary value. The cost of pumping a million foot-pounds is about 1.6 cents with the direct-acting pumps, and 2.5 to 2.9 with high-pressure rotative engines.

Rotary pumps, or those of the crank and fly-wheel type, are not popular among mining men, though a uniform velocity is maintained and the gain in power compensates for the increased initial cost in the proportioning of the engine.

Water-pressure engines are coming much into vogue for utilizing the force from some neighboring stream of water. Reservoirs are established at some distance—400 or 500 feet—above the shaft mouth, and connected by pipes to the engine, the piston of which receives the pressure and communicates its power to the rods of a Cornish pump (Fig. 97); or the engine may be direct acting. In the latter form the hydraulic engine does a fair duty, and as it works equally well under water, cannot be drowned out as can the steam appliance. 280 miners' inches of water falling 100 feet will raise 80 inches 200 feet, or 40 inches 400 feet. Mines having both shaft and tunnel communicating with the surface are admirably convenient to the employment of the hydraulic engines, which may be placed in

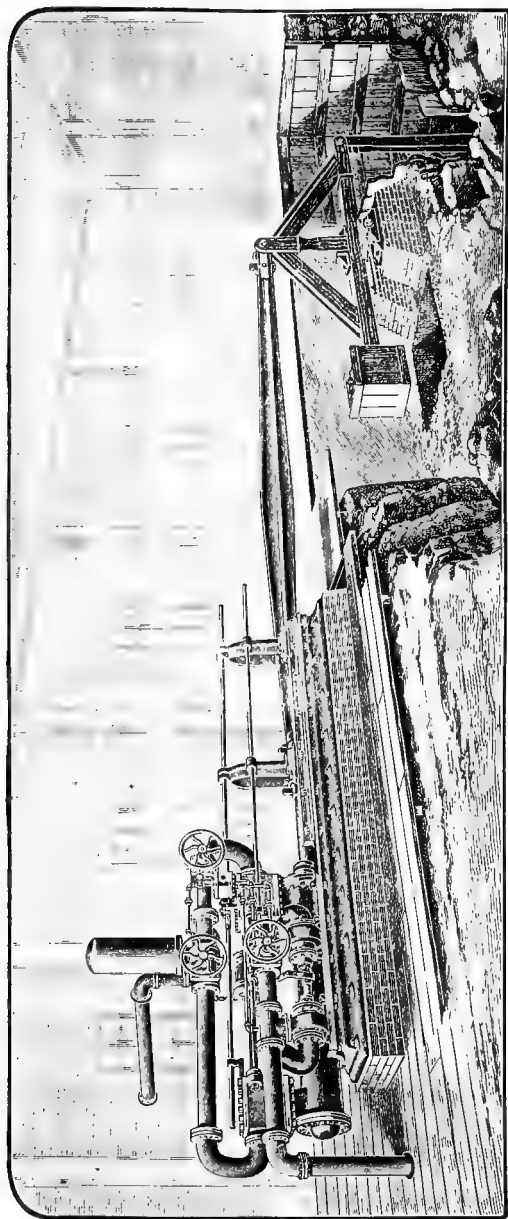


FIG. 97.

the tunnel-level and receive from the surface at the shaft mouth the water-pressure under a great head and discharge it and the pumped water through the tunnel. This is the method at the Comstock mines. The great trouble with these engines is in the valve-gear, which is more complex, and must be carefully balanced, for an incompressible fluid like water, than is necessary for an elastic fluid like steam or air. The cutting off of the valves produces a not inconsiderable concussion in the inelastic fluid which is entering the cylinder at a high velocity, and unless delicately manipulated they fail to operate.

The difficulty in converting electric-motor rotation to reciprocating motion militates against electric pumps for the present.

The windmill has been suggested as a motor for unwatering mines. An experiment on Musquito Pass resulted, three months after erection, in a collection of bric-a-brac for three miles down the gulch.

For limited supply the siphon has done service. One need not dwell upon the principle, except to give the rule. The level of entry must be above the level of delivery, and cannot be over 34 feet below the highest point of the bend; the distance between the delivery and entry may be anything. This takes no account of friction, etc.

A Priestman oil-engine and pump is an admirable self-contained plant for raising small bodies of water through small heights. Being portable, the combination may be moved from place to place as desired.

Ex. 17.—A mine delivers 1900 gallons per minute. Depth of shaft is 465 feet. Required the size of the pump-cylinders under a boiler-pressure of 100 lbs. (gauge) and a $\frac{1}{8}$ cut-off, back-pressure being 16 inches of mercury. Efficiency, 60 per cent.

As the ordinary piston-speed is 200 feet per minute, the discharge of 4.227 cubic feet per second may be delivered at the same speed in the pipe, which is then of a diameter of 15 inches; or if a 10-inch discharge-pipe be employed the velocity therein would be 464 feet. The loss of head would be respectively 1.56 and 11.86 feet. Total head being then 470 or 480 feet, the work of raising the 1700 gallons would be 7,441,500 and 7,599,840 ft.-lbs., respectively. The two steam ends must be capable of 12,402,516 and 12,666,400 ft.-lbs. From

the table on page 122, the mean pressure corresponding to a cut-off of $\frac{1}{2}$ is 0.726 for one pound, and 83.27 lbs. for 114.7 absolute.

Let the average effective pressure be 64 lbs., then the diameter of each steam end should be $12\frac{1}{2}$ inches. Assuming a stroke of 24 inches, each stroke represents 2.53 cu. ft., and the diameter of each water-cylinder would be $10\frac{3}{4}$ inches. If the minimum effective pressures upon the two steam-pistons be taken (25 lbs. per sq. in. on one and 64 lbs. on the other), the diameter of the water-cylinders should not exceed $7\frac{1}{2}$ inches. This discussion neglects the inertia of reciprocating parts.

If the discharge-pipe be assumed at 6 inches in diameter, it would entail a loss of head of 151.47 feet, requiring 475 h. p.

The ratio between the diameters of the two ends of a steam-pump is about 1 : 2 for the smaller sizes, and the steam end is 3 times that of the water end in the large sizes.

Ex. 18.—What volume could be raised by a double-acting steam-pump having water-cylinders 8 inches in diameter, and the steam-cylinders 18 inches, with a 2-ft. stroke? Piston-speed 200 feet per minute. 139.4 cu. ft.

Ex. 19.—What should be the effective steam-pressure to discharge the water, assuming an efficiency of 50 per cent and a shaft 400 feet deep? Assuming a discharge-pipe of 5 inches diameter, p is 83.7 lbs. per sq. in., the loss of head being 88.22 feet.

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CHAPTER XI.

VENTILATION.

41. Laws regarding the ventilation of mines; output dependent upon the hygienic conditions; division of the subject into three branches; the gases encountered in mines, carbonic acid, sulphuretted hydrogen, carbonic oxide, and firedamp; their physiological effects; how evolved, where accumulated, and how removed. 42. Treatment of asphyxiated persons; effect of the gases upon lamps; modes of testing for firedamp; Hepplewite-Gray tester; Shaw's apparatus; explosions; after-damp; influence of the barometric changes upon the evolution of gas; the sole means of obtaining security. 43. Consumption of air by combustion, blasting, etc.; dilution of the products of combustion; volume of air required in a mine for man, light, beast, powder, and extent of working-face exposed; allowance necessary for drag and friction; physical laws of the movement of air. The water-gauge, its use, and the interpretation of the different modes of measuring air; the ventilation paradox. References.

41. FROM an economic as well as a humanitarian point of view, the ventilation of a mine is a matter of very great concern. Neither shaft nor tunnel can be carried more than 200 feet beyond an opening without some special means of stirring and freshening the stagnant air. The men should not be compelled to work in the hot atmosphere of a stope or room vitiated by the variety of gases given off from and by coal, powder, lamps, respiration, rotting timbers, and decomposing ore. These cannot support combustion, nor can they be inhaled with impunity; and such an atmosphere is unfit for respiration, being deficient in oxygen, as well as by reason of the presence of these gases. In coal-mines additional peril accompanies some of these gases, which with the air form

explosive mixtures, the which, bursting into flame, destroy everything in their path and emit dense volumes of poisonous fumes that are fatal to all who have escaped the shock.

Only during recent years does the life and energy of a miner seem to have been acknowledged as having a pecuniary value. The statutes are becoming more and more rigorous in the insistence of safety and hygienic measures. They specify the methods by which the noxious gases may be made harmless, and ordain inspectors with sufficient authority to suggest needed improvements and to punish non-compliance. Not only are the miners benefited by the diminishment of risk, but also the operators, who profit in the increased energy of men working under favorable circumstances. The illumination is better, smoke clears away quicker, and the men are invigorated.

There is a marked contrast between the requirements of vein mines and those of coal or other mines in flat beds, which latter usually have two shafts connected by a labyrinth of workings on nearly one level. The ventilation of metal mines presents by no means the same difficulty as that of gaseous collieries, since as a rule the former class of mines requires but a small supply of fresh air, abundant enough for the health, comfort, and effective work of the men and for the removal of the dead air vitiated by various causes. Trust is placed in natural means of circulating the air by the winze communication of the different levels. This is certainly inadequate, as the British Commission declares that the mortality in well-ventilated coal-mines is less than that of metal-mines, for the ventilation of which it recommends artificial aid. In coal-mines large volumes of fresh air, additional to that required for the men, are necessary to carry off the fire-damp, choke-damp, and other gases both noxious and inflammable, though no serious results may occur from their presence in large quantities. These gases are continually being evolved from the coal and are, hence, an ever-present danger to coal operations, but do not threaten metal-miners. Nevertheless the latter should remember that the inflammable

gases of coal-mines from which they are comparatively free are not the only ones to be guarded against. The cloud of dust formed by blasting the mineral, and the carbonic acid gas, are dangers equally insidious to the health of those inhaling it, which cause as many deaths by what is known as miner's consumption as do the fearful explosions in an equal number of coal-mines.

In addition to the gases found in the normal atmosphere surrounding the earth, there exist in mines several gases which are always the result of decomposition and combustion. Pursuing the nomenclature of early mining, they are designated as "damps." For example, carbonic acid, CO_2 , is known as choke-damp; carbonic oxide, CO , as white-damp; sulphuretted hydrogen, H_2S , as stink-damp; carburetted hydrogen or marsh-gas, CH_4 , as fire-damp or fulminating damp; air vitiated by breathing, having therefore a deficiency of oxygen, as black-damp; and the residual gases of an explosion, after-damp. In metal-mines, carbonic acid and sulphuretted hydrogen are the only gases met with. Both are heavier than air and naturally will be found near the floor of the workings. Coal-mines, however, are troubled with additions, emanations, sudden or continual, of carbonic oxide and marsh-gas, which two, being lighter than air, are to be looked for in the upper portions of the workings, near the roof.

The atmosphere contains usually in every one hundred grains 76.84 grains of nitrogen, 23.10 grains of oxygen, 0.06 of carbonic acid, occupying respectively 79.02 per cent, 20.94 per cent, and 0.04 per cent of the original volume. These gases occur not in chemical combination, but as a mechanical mixture from which the oxygen may be extracted under given conditions. The oxygen, which is colorless, odorless, and transparent, unites with all other chemical elements and forms compounds; and all the ordinary phenomena of light, heat, and fire are the result of the union of other elements with oxygen. This active principle is essential to life and to all other processes of combustion, and when combined with

carbon forms **choke-damp** or **white-damp**; with pyrites, a common mineral in the coal, sulphuretted hydrogen; and with marsh-gas, water or steam and carbonic acid. An atmosphere deficient in oxygen will no longer support combustion, and a lamp or flame immersed in it is extinguished; so, too, air-breathing animals are suffocated when the percentage of oxygen in the air is below 15.

Nitrogen, the predominant constituent of unpolluted air, is colorless, odorless, and incapable of supporting combustion or animal life. It is inert in its effect upon the system, but extinguishes flame immersed in it.

The carbonic acid, which is an insignificant constituent of the atmosphere, is, however, to be expected in all abandoned and unventilated places. It is given off in quantity by the respiration of men and animals, by the combustion of lamps, fuel, explosives, timber, and organic matter, and by all substances in a state of decay or fermentation. It, also, is colorless, odorless, and at a normal pressure of 30 inches of mercury and a temperature of 32° F. weighs 128.45 pounds per thousand cubic feet, as against 81 pounds for an equal volume of air under the same conditions. It is easily detected, for, being incapable of sustaining combustion, light burns dimly when immersed in it. When, therefore, the gas is observed by its action on a flame, the atmosphere is at least unfit for respiration. The gas has an injurious effect upon the human system. Air containing 2 per cent of it produces an overwhelming depression upon those breathing it; with 6 per cent lights are extinguished; and with 10 per cent it is positively fatal by producing suffocation. As it accumulates in the lower part of idle workings and where the air is stagnant, no one should venture into abandoned works without having previously tested by a flame the condition of their atmosphere, and if the light is extinguished, the poison must be swept out by a strong current of air, which, in shafts, may be incited by rapidly raising and lowering a bundle of hay which then is ignited. Small volumes of carbonic acid may be removed from the atmosphere by the use of absorbents

like lime or ammonia. Its presence may be chemically detected by the milkiness produced in a test-tube containing baryta hydrate.

Sulphuretted hydrogen, which at 30 inches barometric pressure and a temperature of 32° F. weighs 94.62 pounds per thousand cubic feet, is an extremely poisonous and common gas occurring in mines. As it is the result of the decomposition of pyrites, always accompanied by heat, this gas is a warning of incipient fires. In the gob and abandoned portions of the mine, the pyrites and coal-waste subjected to pressure from the roof, and in the presence of moisture, decompose with the development of heat and possibly flame, of which the presence of sulphuretted hydrogen is an important indication. Being colorless but strongly odorous, its presence is readily detected. It does not support combustion, but is itself inflammable. A flame will burn in a mixture of it with air, but is extinguished in an undiluted atmosphere of this noxious gas, which in this state is fatal to life.

Carbonic oxide, which is claimed to be a normal gas existent in coal-mines, weighs 78.305 pounds per one thousand cubic feet at a temperature of 32° F. and a barometric pressure of 30 inches. This gas has neither color, taste, nor smell, and is exceedingly poisonous. One half per cent of this gas in air renders the atmosphere fatal to life if breathed for ten minutes. It acts upon the system by combining with the oxygen absorbed in the blood, forming a stable compound, reducing the hæmoglobin, and insidiously and surely destroying the blood and tissues. As carbonic oxide is an unstable compound usually resulting from the imperfect or secondary combustion of a gas or of carbon in the deficiency of oxygen, the existence of this gas in mines is doubted. It is only barely possible that it might be found in the goaf, where the oxidation of pyrites and the absorption of oxygen by the fine coal-dust may have depleted the air of its vital element, while thereby giving rise to a sufficiently high temperature to incite combustion with the development of carbonic oxide.

Light carburetted hydrogen or marsh-gas is a stable, never-

failing constituent among the products of dry distillation of organic matter, and exists as the predominant constituent of the compound gas known under the general term of fire-damp because of its ready ignition by flame with a mixture of air. Its weight at the normal barometric pressure and a temperature of 32° F. is about 45.22 pounds per thousand cubic feet. It is absorbed in the coal, diffused through its pores, collected in crevices or cavities, and even stored up in reservoirs, having been exuded from the coal in the early stages of decomposition of the organic source of the coal, or expressed during the geological movements of the earth's crust. It is not a constituent of the coal, but is entirely free from chemical combination with it, and continually exudes with greater or less violence; it is liberated in volumes by falls in the mine-roof, by squeezes or creeps, and by any sudden fall in the barometric pressure. The deep portions of coal-seams appear to be more heavily charged with this compound gas than are workings in coal-beds so close to the surface as to allow of its escape. No coal-seam should be regarded as free from liability to irruptions of this gas.

Most coals contain this gas enclosed or occluded in their pores, so it must not be confounded with the gases chemically compounded with the coal. It is under high pressure, and is given off into the workings of the mine with greater or less violence from the fissures or crevices, sometimes without warning, sometimes accompanied by the heaving of the floor and the trembling of the roof, but always more or less distinctly audible. The volume so emitted from the coal varies in amount from 9 to 100 cubic feet per ton. From anthracite the discharge is most copious, and from bituminous coal the least.

Frequently it escapes into the mine without warning; its presence is not always detected or manifested, as it accumulates in a nook or under a platform until some untoward circumstance brings it into contact with a naked light. The slight hiss accompanying its exudation is hardly enough to be distinguishable. These "blowers," of all sizes, up to the

outbursts that for a time overpower the ordinary ventilating current, contain 90 per cent of marsh-gas, and may be liberated anywhere and at any time. For a long time it has been recognized as a constituent of the gases entering mines. It escapes at various springs and salt-mines; it has fed the sacred fires of Baku and the mud-volcanoes of Bulganak; it has been found in the Silver Islet mine, in the iron-mines of Alsace, and in the lead-mines of Tuscany; and some years ago an explosive gas was met with in driving the lake tunnel at Chicago. It constitutes from 40 to 90 per cent of the natural gas, and is obtained among the volatile and combustible constituents in the ultimate analysis of coal. While sinking shafts through porous strata fire-damp has been encountered, and precautions are therefore necessary in regions of natural gas, or in formations closely above the coal horizons.

Measurements of the pressure of this gas have been made by the Royal Commission on Accidents in Mines, with the result that it has been found frequently to be as great as 450 pounds per square inch. Sudden outbursts may therefore be expected when escape is afforded, and these, according to the treatise of M. Roberti Linterman, preponderate in coal-seams disturbed by faults, foldings, or thinnings-out, and are influenced by the dip of the seam. They occur without any premonitory symptoms and even in districts heretofore free of gas. They are most voluminous in the periods of pillar-robbing or during the exploration of virgin ground. The disengagement of gases increases both in intensity and frequency with the depth of the workings and with the presence of permeable masses surrounded by rock-masses so hard and compact as to constitute effective barriers against fire-damp. In every district are found one or more infested zones of gas.

Owing, therefore, to the inevitable occurrence of this inflammable gas in all coal-seams, and the uncertain quantity which may be thrust into contact with the flame of the illuminating lamp or of the explosive employed in the mine, fire-damp is the dread enemy of coal-miners. The amount of gas which renders the mine unsafe cannot be stated, for

while an atmosphere containing two per cent or more is neither injurious to life nor dangerous in mines, nevertheless its presence to that extent in the air discharged from the mine into our atmosphere indicates a probably excessive accumulation in some of the workings. The Coal Commission of Austria, in its classification of mines according to the composition of the air at outlet of their ventilating-shaft, regards those having more than 2 per cent of fire-damp and carbonic acid in their return-air as fiery, and those having less than one per cent of gas-admixture as safe.

42. The effect of the admixture of the mine-gases with air upon life or upon an illuminating flame differs somewhat. In an undiluted state each and all extinguish flame and do not sustain life, but when mixed with air the carbonic oxide and sulphuretted hydrogen are poisonous, while at the same time they support combustion. The marsh-gas is inert in its influence upon life, but is capable of ignition; while the carbonic acid is depressive in its influence both upon flame and upon life.

Those which burn with a dilute mixture of air give evidence of their presence upon a flame immersed in it by the elongation of the flame surrounded by a blue nimbus or aureole. The more volatile the illuminating-oil used in the lamp, the more sensitive is the flame to the presence of these gases. Thus naphtha, benzine, and kerosene, in the order named, are far more sensitive indicators of gas than is the heavy lard-oil. Admixtures of these gases with air will burn freely, but when the proportion of gas reaches a certain specific amount the ignition may take place rapidly, and if the products of combustion cannot escape equally rapidly, explosion ensues, the force of the explosion and the dangerous degree of dilution of gas varying with the different gases.

The range of proportion of gas dilution between the lower and the upper explosive mixture is least in the case of fire-damp, which may vary in amount between 5 and 13 per cent of the total volume of air; is greatest in the case of carbonic oxide, which will explode with any mixture containing

between 13 and 75 per cent; while with sulphuretted hydrogen the variation lies between 9 and 28 per cent of gas in the mixture. The gases are mentioned also in the order of the decreasing danger of explosibility, the first offering the greatest risk. When the atmosphere in which a flame is immersed contains a percentage of gas approaching the explosive limit, the cap and nimbus become large, and the flame almost invisible. The rapidity with which the ignition is propagated depends upon the nearness of the proportions of the admixture to the figures given, and when the maximum explosive ratio as indicated above is reached, the propagation is instantaneous, and the concussive force of the explosion is also a maximum; as the percentage of gas recedes from these ratios or increases beyond the limiting explosive proportions, the violence of the explosion decreases. When either of the gases is undiluted with air, the light placed in contact with it is extinguished. In other words, a flame may be immersed in the workings filled with fire-damp; and if the line of demarcation between the air and the gases is "sharp," no explosion will ensue, but the flame will enlarge and, after a little fluttering, become extinguished. This is equally true of the other gases, though the percentage of their accumulation in mine-workings is so low and their diffusion so perfect that no undiluted accumulation of them is likely to ensue.

In their pure state the gases are extinctive of flame and, of course, destructive to human life. Diluted with air only carbonic acid and carbonic oxide are injurious to life—the former when in proportion of 10 to 90, and the latter when comprising but one-half per cent of the mixture. There are hence two methods of discovering the probable unsuitability of air for respiration—the extinction of flame by an excess of carbonic acid, and the flame aureole from the combustion of carbonic oxide, the odor of sulphuretted hydrogen being a sufficiently strong warning without other index of its noxious presence. Fortunately, however, the presence of carbonic oxide need only be feared after a fire-damp explosion or after the blast of one of the lower grades of black powder.

The residual gas "black-damp" is extinctive of flame because of the deficiency in oxygen extracted by the combustion of carbon, and when pure is theoretically composed of 52 per cent nitrogen and 48 per cent of carbonic acid. The percentage of a mixture of oxygen, nitrogen, and carbonic acid in the atmosphere that is extinctive of flame is almost identical with that of the air expired from lungs of men. Dr. F. Clowes, as the result of a series of experiments in relation to the lighting of mines and the behavior of lamps, has ascertained that the percentage composition of the residual atmosphere in which flame was extinguished is oxygen 15.7, nitrogen 81.1, carbonic acid 3.2, while that of the average exhaled air is 16.15, 79.9, and 3.95, respectively. But in an atmosphere that would just extinguish flame the miner need have no occasion for alarm unless the lamp has been raised into a stratum of some inflammable gas. Until the atmosphere contains as much as 8 per cent of carbonic acid the air does not seriously affect the breathing. With that degree of saturation the inhalation quickly produces unconsciousness and may soon prove fatal. In addition to that just mentioned, the other common causes affecting the suitability of air for supporting life are the reduction in the proportion of oxygen below 12 per cent and the presence of carbonic oxide to an amount exceeding 0.2 per cent.

In an atmosphere deficient in oxygen or possessing deleterious gaseous elements, the action on the system is that of a gradual impairment of all functions, and is first manifested by a loss of consciousness followed by a very marked panting in the effort to supply oxygen to the lungs, and accompanied by increased pulsations of the heart. The evidence of impoverished blood is soon apparent in the face, which assumes a leaden color. The countenance then becomes swollen, the features are distorted, sometimes the eyes protrude, and the face often assumes a livid color; and as asphyxia is pronounced, there is a sudden cessation of the action of the senses, of the pulsations of the heart, and of the respiration.

Persons asphyxiated by any of these gases may be revived by blowing oxygen into one of the nostrils, the other being closed, and by inducing artificial breathing. Epsom salts, and water acidulated with vinegar, are better than alcoholic stimulants. The warmth of the body should be kept up, and mustard plasters applied over the heart and around the ankles. If these produce no effect, recourse must be had to blood-letting from the foot or jugular vein, and, as a last resort, an opening into the trachea, by which pure air is forced into the lungs.

Those overcome by the inhalation of carbonic oxide can be resuscitated only by a prompt and copious supply of pure oxygen to the lungs.

42. There seems to be no means of preventing the evolution of the gases into the mine or the numerous causes resulting in disaster from their ignition. Various devices have been applied seeking to escape the consequences of this inflammable and explosive agent; but none has been found feasible other than a thorough system of ventilation which will dilute and "drown out" the enemy.

A copious supply of air which will dilute the gases to a safe, healthy limit of inhalation is imperative, as is also the maintenance of this current through air-ways of cross-sectional area ample to convey it at a velocity of less than 300 feet per minute in mines employing the common Davy or Clanny safety-lamps, and less than 800 feet per minute in mines where the bonneted forms of lamps are used.

The enforcement of a rigid discipline and of the regulations regarding safety-lamps and the use of blasting agents, as well as of the time of blasting, will contribute to immunity from gas-explosions, but there is no means of correcting the wilfulness of the miner who endangers life and property in order to steal the luxury of a forbidden pipe.

When fire-damp has been allowed to accumulate to a dangerous degree and is fired by an imperfect lamp, a naked flame, or the flame of a blasting agent, and is exploded, it is rare that doors withstand the shock, and if no safety-traps are

sprung down (see 53) in their stead, the air-current is deranged, and the men have no chance for their lives in the "after-damp." This residual gas theoretically contains seven parts of nitrogen, N, two parts of steam, and one of CO₂, without any oxygen—a mixture wholly incapable of supporting combustion, certainly one irrespirable, as the condition of the unfortunate victims certifies. In practice, however, the exact composition of the after-damp depends upon the composition of the explosive mixture, for though in every case the fire-damp would be entirely burned, the amount of oxygen present may not be sufficient for complete combustion, and carbonic oxide becomes an important and copious element in all after-damp.

In its production two volumes of marsh-gas (CH₄) combine with 19 vols. of air and develop 23,550 heat-units, giving a temperature of 6064° F. (6524° absolute), and a pressure of 179.3 pounds per square inch. If *m* be the weight and *c* the specific heat of a gas, the heat required to raise it *t*° is expressed by *mtc*. To raise the 2.75 pounds of CO₂, 14 pounds of N, and 2.25 pounds of water from 52° F. to *t*° F. requires

$$2.75 \times 0.1711t = 0.470t;$$

$$14 \times 0.1740t = 2.418t;$$

$$\text{and} \quad 2.25(160 + 990) + 2.25 \times 0.2675t = 0.602t + 2587;$$

$$\text{whence} \quad 0.470t + 2.418t + 0.602t = 23550.$$

To this gas is attributed the loss of many lives in a mine-explosion—more, in fact, than are the result of its concussion or of contact with its flame. The perfectly natural appearance of the body, lying often by a lamp still burning, proves the cause of death to be some insidious poison which is combustible. All members of rescuing parties entering the workings thereafter should take due precautions against the inhalation of this carbonic oxide gas.

The force of the explosion of two volumes of marsh-gas developing 23,550 heat-units may be ascertained to be nearly 30,000 pounds per square foot by the following analysis:

Assuming the initial temperature of the marsh-gas and the mine air to be 62° F. or 523° absolute, because it requires 3,534 heat-units to raise the aggregate products of combus-

tion one degree, the degrees to which the final gaseous products of combustion will be raised are

$$23,550 \div 3,534 = 6663^{\circ} \text{ F.}$$

The volume which these products seek to attain is

$$(523 + 6663^{\circ}) \div 523 = 13.8 \text{ atmospheres.}$$

13.8×14.7 per square inch equals 29,211 pounds per square foot.

An attempt has been made to hypothecate a relation between the periods of gas outbursts and the movements or seasons of low barometer, but the author fails to find any simultaneity in the phenomena. A falling barometer has not invariably been followed by a heavy discharge of gas, nor does a study of the tables show its unfailing precedence to the evolution. Whilst laying stress on the acknowledged fact that December is the worst month, there appears to be no "off day" for explosions, which are equally abundant on any day of the week. An excessively low barometer at the sea-level is 28.3",—a fall of only 6 per cent of the total pressure, and of but 1 per cent, or less, of the pressure of the magazine gas. Upon the emissions from the pores of the coal and from goaves the effect of a barometric depression is noticeable. But even here an acre of ground of standard thickness will, with a barometric fall of 0.1 inch, exude only 18 cubic feet of mixture for every 25 yards of length of face exposed.

A tabulation of the barometric variations with reference to mine-explosions was made in Westphalia, during 1896, with the result that of the 42 explosions recorded 41 are attributed to fire-damp alone, while in one of them coal-dust participated; out of the total number, 27 happened when the air-pressure showed a tendency to fall suddenly or was at its minimum, while in the 15 others the air-pressure was maximum, or showed a tendency to rise. As regards the places where they occurred, the explosions are divided into eight in exploring or preparatory workings in rock, 26 in preparatory

workings in coal, and eight in the actual getting of coal, while the gas issued slowly in 28 and suddenly in 14 cases.

It is fortunate that the various gases evolved from coal or produced by the various processes of decomposition, combustion, and exhalation do not accumulate in separate layers in the workings with the heavier gas at the bottom and the lighter one near the top, except in abandoned places where the air is allowed to stagnate; but instead of this even a little circulation will set up an individual motion of the separate particles of the gases by which they become gradually diffused throughout the mass, until, after sufficient time has elapsed for the purpose, they are found intimately blended, whatever may be their relative densities. This is not a chemical mixture, but a purely mechanical blending, depending upon the relative tensions of the gases. The rapidity of this diffusion into atmospheric air is inversely proportional to the square root of the density of the penetrating gas. Marsh-gas, therefore, mixes most readily with the air, carbonic oxide not quite so readily; sulphuretted hydrogen less so, and carbonic acid making with difficulty an intimate mixture with air. This principle of diffusion is an exceedingly valuable one to the safety of the mine and the purity of its air, since the more ready the diffusion of the gas the more easily will the gas be cleared away. Thus, by the creation of an air-current throughout the workings, the gases are mixed with the circulating pure air, are diluted, and swept out of the mine.

The condition of the coal workings is usually examined and tested daily by the fire-boss, one of whose duties consists in ascertaining the degree of saturation of the mine air at every place of work before the men are permitted to enter. The test is made by a candle or safety-lamp, the flame of which gives evidence of the presence of an accumulation of combustible gas. This method requires a skilful, steady hand and considerable nerve. Shading the flame of a candle or lamp with one hand, and raising it upward, the fire-boss watches the behavior of the light. If any inflammable gas is present, the flame elongates and becomes smoky. In this

event the test ceases, the flame is lowered, and the fire-boss withdraws. The lamp is raised upward because not only is the tendency of gases to explode greater when fired from below than when the flame is applied on top, but the stagnant gas always gathers near the roof. The face of coal or the room showing these symptoms of danger is then supplied with more air, the employes being meanwhile barred from entry to the place.

The ordinary testing-lamps do not reveal the presence of a quantity of gas less than 2 per cent. The Hepplewite-Gray detects smaller quantities than does the unbonneted Davy or Clanny. It burns benzoline, and shows a cap $\frac{1}{2}$ " high in the presence of 1 per cent of CH_4 . The Pieler spirit-lamp is always a good gas-tester, which in air containing $\frac{1}{2}$ per cent of combustible will give evidence of it in a cap one inch high. The Wolf safety-lamp, burning naphtha, shows a very conspicuous halo when placed in a mixture of fire-damp and air; in a mixture of $\frac{1}{2}$ per cent of gas the flame is $1\frac{1}{2}$ inches in height, and at $2\frac{1}{2}$ per cent gas the flame is broader, and may even be extinguished. A very sensitive gas-detector, much used in England, is the one described in the Transactions of the Mining Institute of Scotland, vol. VIII, page 3. The Shaw gas testing-apparatus, which recognizes the presence of a minute quantity of gas in the air and is used for the approximate test of mine air, meets with favor where the exact analysis of the mine atmosphere is not required. While the apparatus is capable of demonstrating the quantity of explosive gas in the atmosphere, it is incapable of distinguishing between them, and thus fails to furnish any clue as to the variety of gas therein contained. Various other fire-damp detectors have been placed upon the market, but none have demonstrated their reliability. Those depending upon the difference in density of the gases are unreliable, because changes of temperature will produce similar results. Aitken's indicator is ingenious, but not much more reliable. Its thermometer is coated with platinum-black and plaster of Paris, and when exposed to fire-damp becomes heated. If the difference of

temperature between it and the normal air always bore a comparable ratio to the percentage of fire-damp contained, it would work well. One objection to the special forms of gas-detectors is, that they do not serve for illumination, and a lamp must also be carried.

43. In attempting to specify the amount of air required for proper ventilation of a mine, we are treading upon uncertain ground. Within close limits we may ascertain the amount required for the vital chemical purposes of horse, light, and man—constant sources of vitiation. A pound of carbon requires for complete combustion $2\frac{3}{8}$ pounds of oxygen, and produces $3\frac{3}{8}$ pounds of CO_2 . Hence the ordinary-sized mining-candle burns up 11.8 cubic feet of air, and discharges 3 cubic feet of CO_2 . Eminent medical authorities state that a man consumes about 1 cubic foot of air per minute, converting the life-giving principle into 2.1 cubic feet of CO_2 per hour. The respiration of a horse is about 13 cubic feet CO_2 per hour. The deflagration of a pound of explosive produces about 2.6 cubic feet. According to Angus Smith, two hewers using a $\frac{1}{2}$ -pound candle and 12 ounces of powder produce $25\frac{1}{2}$ cubic feet CO_2 in a shift.

The amounts of air sufficient to satisfy the conditions of combustion during the generation of the respective amounts of CO_2 are small, and if the exhalations were instantly removed, the theoretical chemical supply would suffice. But the air in the confined spaces of mine-workings is somewhat stagnant, and the atmosphere is further deteriorated by the effluvia from man and beast. Some of these are not easily detected chemically, but are more deleterious than CO_2 , which is not the sole test of vitiation.

To sweep away the hot, noisome emanations, the poisonous exhalations, the unconsumed azotic gases, and, finally, the exuding pent-up gases from the coal, and to render them comparatively harmless, require a very large volume of air for their dilution and renewal. Pure dry air contains, by volume, 21 per cent of oxygen, O, and 79 per cent of nitrogen, N; and every 1000 cubic feet of it, weighing nearly 81 pounds, con-

tains only about 18.7 pounds of the life-supporting constituent, the remainder being matter inert in its physiological effects.

Judging by the rough and unsatisfactory test afforded by the sense of smell, the air of a room ceases to be good when it contains 8 parts of CO_2 in 10,000. And to preserve the lowest standard tolerated by sanitarians, 1 in 10,000, the supply will be proportioned as follows: 59 cubic feet per hour per light; 4585 per horse; 9192 per pound of powder; and 1500 per man employed. Competent writers vary in this matter, and the statutes of the various States differ in their requirements (55 to 300 cubic feet per man per minute). But the allowance for a mine cannot be based on the single per capita element, for it will be seen later that the friction or "drag" of air, in moving through headings and along faces which increase with the developments, diminishes the volume of air actually allowed to move. Moreover, the emission of gas from the strata, proportional to the area exposed and the character of the coal, constitutes another and constant source of pollution. In all preparatory and prospecting work in virgin ground an extra allowance of fresh air is necessary. Cognizance must be taken of this unfailing source to the extent of an hourly allowance of 0.3 cubic foot of air per square foot of working face, in a dry, dusty, fiery mine. For a non-gaseous seam 0.1 cubic foot will suffice. Some property-owners allow also 200 cubic feet air per hour for every acre of goaf. For the eruptions from the magazines no provision can be made, except vigilance and discipline.

The water-gauge (Fig. 98) consists of a U-tube, whose arms contain water and are provided with measured scales graduated to inches above and below the normal of the water in the columns. If such a gauge be inserted through the door or stopping separating the bottoms of two ventilating-shafts of the mine, the water will remain at a normal level, if the temperatures and tensions of the gas and

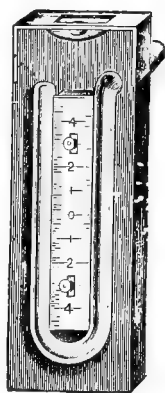


FIG. 98.

air in the two shafts be equal; but if by any cause the tension or temperature be changed in either shaft, the ensuing difference in pressures will be communicated to the connecting-arms of the water-gauge in such manner that the cool or denser air will force down the water column in the arm on its side of the stopping and elevate the water column on the opposite side. This difference in level, m , is read on the attached scale and represents the motive column, M , which is capable of producing motion. If, now, this excessive pressure may be allowed to expend itself in producing motion through the workings of the mine, in circulating the air which ultimately is discharged through the lighter column of upcast, the level of the water in the gauge will fall slightly until equilibrium will be established, when its difference in level will represent the difference in pressure, p , at the bottom of the two shafts on either side of the stopping. The excessive pressure, P , is expended in doing work of two kinds: (1) in overcoming the friction to the passage of the current of air through the workings, from one shaft to the other, and (2) in creating motion. The latter work is measured by the velocity of the outgoing current, the former is measured by the height of the water in the gauge—in other words, the manometric depression, m . Its difference in level constituting the water-gauge reading measures the force which is required to drive the air through the mine. It measures the loss due to friction or the “drag” of the mine. Be the quantity of air large or small, it gives no measure of that volume, but, paradoxical though it may seem, only of the power of the ventilator. The resistance of the mine is a definite quantity, and bears no relation to the capacity or qualities of the ventilating appliances or methods. The water-gauge, therefore, which measures this resistance is a “function of the mine,” and by it may be determined the relative efficiency of the mine to pass air through its ways. The water-gauge reading in the majority of mines varies between 1” and 3”. Few have a larger resistance. The mine with air-ways of large cross-sectional area and with a well-distributed current should have a

water-gauge reading, or resistance, not exceeding one inch for each hundred thousand cubic feet circulating through it. The mine having a larger ratio of water-gauge reading than this either has air-ways of insufficient size or is not receiving the current properly regulated. The resistance of the mine to the passage of an air-current is often expressed in the term, the equivalent orifice of the mine. By this is understood the area of a thin orifice which offers a resistance to the passage of a current of the same volume, Q , equal to that which is circulating through the mine. The equivalent orifice of the mine, A , bears a certain ratio to the quantity Q , and to the water-gauge reading, m , which is variously expressed by different authors, the general formula being

$$a = CQ \div \sqrt{m},$$

the value for C varying between 0.00037 and 0.00066. Q is the volume in cubic feet per minute.

The equivalent orifice of most mines varies between 10 and 100 square feet. Inasmuch as water is about 833 times as dense as an equal volume of air, the column depressed in the water-gauge corresponds to a height of $833m$ inches of air at 62° F. and 30 inches barometer; the head, M of air measured in feet, to which the manometric depression is due, is

$$M = 69.4m.$$

The value for the corresponding differential pressure, P , in pounds per square foot, is

$$P = 5.184m.$$

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CHAPTER XII.

METHODS OF VENTILATION.

44. Methods of ventilation of a tunnel or advancing gangway; by conduit or brattice; single- and double-entry, and outlet; diagonal, or adjacent, systems for double-entry; increase of temperature with depth; limit of the depth of mining; natural method of ventilation by two outlets at different levels; limitations of the method by season and depth; ventilation of railroad tunnels; account of the different experiments and that finally adopted. 45. The flow of air by changes of pressure or of temperature; the flow of any fluid under a change of tension; motive column; formulæ. 46. Methods of accelerating natural ventilation, etc.; furnace ventilation; cost and construction of the furnace; temperature and volume of the air produced; dangers and limitations in its employment; dumb channels in fiery mines; exhaust-steam as a ventilator. 47. Mechanical ventilators; description of hand-fans and their adaptability; blowers; Root fans; champion blowers; use of compressed air as ventilator; exhaust-fans; details in the construction, arrangement, efficiency, and cost of the same; Guibal fans; lines of improvement; method of housing; outlets and connection; description of the Waddle, Schiele, Lemielle, Cooke, and Fabry fans; comparison of them; effect of a low barometer and high temperature on the volume of the exhaust; fan *vs.* furnace. References. 48. The theory of the action of the fan; its equivalent orifice; its efficiency. 49. Principles of design for fan; formulæ; example. References.

44. To secure ventilation in the confined workings of a mine, a conduit must be furnished by which the warmer and lighter air may ascend to be supplanted by cold or compressed air entering by a different compartment; and to maintain a constant air-current throughout the workings both inlet and outlet must be afforded for the flow by means of two separate entries or by partitions in the one shaft. Shafts, in process

of sinking, or a mine having but a single entry, may discharge their vitiated air through the wooden air-tight box-pipe provided for the purpose, or, if there is small liability of corrosion, through a galvanized iron pipe, the remainder of the entry furnishing the inlet. Because of the wide difference in the areas of the two air-ways so provided, the ventilation is not likely to be good, and it is far better to divide the main tunnel or shaft or mine working into two compartments of nearly equal area, one of which will serve as an outgoing conduit.

From the fact that the current in a single-entry mine is continually interrupted by the other uses to which the compartment is put, and that there is a liability to injury of the partition, box or pipe, this plan is objectionable when a large volume of air is required, because the safety of a great number of men is dependent upon this airway for escape. The wind, moreover, disturbs the ventilating current; the movement of cars, cages, and rock or coal in chutes is also irregular in its influence upon it; and the unusual heat from underground steam-pipes, engines, etc., sets up counter currents, though any of the causes mentioned may occasionally have a beneficial effect. Thus a double-entry to the mine becomes not only precautionary, but also imperative; and as the depth and extent of workings increase, the insufficiency of a single-entry becomes more and more manifest. Even metalliferous mines should be provided a double-entry, for the numerous caves that have occurred, penning in dozens of men without chance of escape unless the rescuers can reach them before suffocation ensues, and the fires that frequently cut off the employees from the outlet and suffocate them before extinguishment is effected, are sufficient arguments in favor of double-entry, even if the necessities for better air do not appeal to the operators.

The coal-mining ordinances now exact two distinct outlets, separated by a safe distance of unbroken rock. The upcast, advisably, should terminate in a large chimney, high enough that its draught be not influenced by changes of wind

or the surrounding buildings. The location of the two entries, in reference to each other, varies within wide limits. One plan consists in having them near together, thus concentrating the plant. Both air-ways being carried with the development, the current passes through to the extreme end of one and return by the other. Then as the work deepens, each lower lift is connected with the air-ways of the upper lift, and receives ventilation with its advance. The other plan is the "diagonal system," the shafts being at the extremities of the workings. While this is well enough for long-wall method, the ventilation must meanwhile suffer until the connection has been made.

Two compartments in a single-entry may be easily obtained in coal furnishing sufficient rock from the roof or from partings by driving a wide gallery and walling it up centrally with the waste; but if there is not rock enough for this, two entries are carried, with the usual pillar between them, having connecting "throughs" at intervals of less than 100 feet, each being closed as fast as the next one is completed. To ventilate that part of each entry between the last connection of the entries and its face, it is subdivided by a canvas brattice along which the current moves (see 53), fastened at the near side of the "through" and leading up to the work; or the faces may be connected by pipes through the door closing the intake entry without interfering with the haulage. The practice of relying upon diffusion to do the work of ventilation is pernicious. These remarks also hold true regarding the "throughs" connecting the rooms in pillar and stall working, where diffusion is usually relied upon for the needful amount of oxygen.

A large number of mines, even some of the coal-mines, depend for their ventilation solely upon natural means, and this may suffice in small mines. But as the workings are extended the numerous connections which are necessary for development or convenience of handling the materials may be planned to serve also for ventilating ways without additional cost.

In planning the direction of gangways and of rooms in coal-mines, usually the question of haulage is of the first consideration, unless it be that the "cleats" are so pronounced as to determine the direction of work. At the same time due attention must be given to the matter of ventilation, that the requisite amount of air be given each working-room, and that too many men be not dependent upon the same air-current circulating through the mine; whenever the mining conditions require a subdivision of the incoming air-current into small currents, each being distributed to its own district and group of men and each separately discharged, it becomes evident that the ventilation of such gaseous mines must receive special attention, not only as to the direction in which the air-ways are driven and their cross-sectional dimensions, but also as to the means of producing the supply of air. In such cases the fresh air should be carried, if possible, to the deepest point in the mine, whence an ascending current may be conveyed through the workings until it is returned to the surface. Especially is this advisable in steep coal-seams carrying fire-damp.

The ventilation must be so arranged that as many independent ventilation districts as possible be provided with separate air-currents; and especially must each lift of workings be supplied by the shortest way with the necessary quantity of fresh air, while within the separate lifts of workings the air-current must always be ascending—except in cases in which the descending air-currents are not used for any further ventilation purpose, or when, in certain well-ventilated working places, great thrust of the measures renders very difficult the keeping up of special return air-ways.

In metal-mines, where the development is of slower growth, the rock hard, and a comparatively small force is at work, the amount of air required is small, either for inhalation or for the dilution of the gases developed therein; hence a single shaft with two compartments may suffice, the circulation being left to natural sources. This, however, will be inadequate when the shafts and workings reach a depth of several hundred feet, in which case other means must be em-

ployed. The use of compressed air for drills, pumps, etc., may supply the deficiency of pure air which natural ventilation may fail to furnish, yet a fan, exhausting the air from one outlet or forcing the air into the other, seems imperative with extensive workings.

Below a moderate depth, where atmospheric and surface changes cease to have influence, there is in the undisturbed rock an increase in temperature with an increase in depth. The depth at which the temperature of the ground will be found to be invariable and equal to the natural temperature of the locality is about 50 feet below the surface. Beyond this it is an observed fact that in all artificial openings the temperature of the rocks increases for at least a moderate depth, within which the mine operator is concerned, at the rate of about one degree F. for every 68 feet of depth. This increment is not constant for all localities, nor indeed for the same mine, but generally it may be said that as we go down the temperature of the mine increases more or less uniformly. This increased heat is often a great drawback to mining, and will ultimately limit it apart from the lesser mechanical difficulties. As to what would constitute the limiting depth to which mining may be prosecuted, it can but be said that at present several mines, with the exception of the Comstock and those which are in ore-bearing districts feeling the effects of solfataric action, are working at over 4000 feet. Regarding the exceptions stated, it is certain that unless some means be discovered for rendering their lower levels habitable, the limit of mining depth is soon reached. It is stated that a 2800-foot level of the Yellow Jacket Mines has been abandoned because of the excessive temperature, in many rooms of which the miner is compelled to return to a cooling station after laboring only twenty minutes.

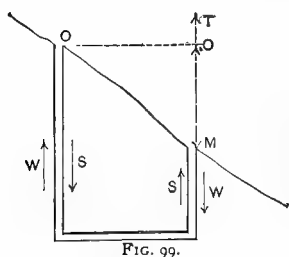
An interesting report bearing upon this question of the rate of increase of temperature with the depth of subterranean explorations, made by a sub-committee of the Royal Commission on Coal, reaches the following conclusion: That the limit of depth to which mining is possible depends upon

human endurance of high temperature, and to the extent to which it would be possible to reduce the temperature of the air which came in contact with the heated rocks; that there is no limit caused by considerations of a mechanical nature as to the size of rope for hoisting-engines, nor by any consideration of the enhanced expenditure for shaft sinking, for haulage, or for pumping; regarding the latter, the experts testifying before them demonstrated that water is seldom, if ever, met with in large quantities at great depths in mines. It therefore appeared that this increase in temperature is the only element needing consideration regarding the limits of prospective sinkings or workings.

A summary of the results of temperature observations made under the direction of the British Commission Committee shows the mean increase of temperature per foot to 0.01563, or one degree F. in 64 feet, the extremes being 0.0077 in the Bootle waterworks bore-holes, and 0.025 in the Carrickfergus shaft. At the Adelbert shaft, Prussia, observations five times a month, in different levels, for a year could deduce no regular law of increase; at the 30th level, 3200 feet, the temperature was 98° F.

The "natural ventilation," so-called, depends upon the foregoing principle, that the relative temperatures of the air outside and of that inside the mine are such as to give rise to a change of volume and of tension that will incite a circulation.

So that if two openings be made and connected below, a current will be established down the lower and shorter opening in winter, and up the same during the summer, as the arrows (Fig. 99) marked *S* indicate. In winter the direction of the current follows that of the arrows, *W*. The amount of air thus set into circulation by the changes of the exterior temperature will depend upon the relative difference of temperature between the mine and surface, and also upon the depths of the shafts. When these



differences are slight it is not easy to predict the direction which the current will take. As for example, in the fall and spring it will fluctuate from one to the other. When, however, these differences are great, a current will be set up which tends to continue in the same direction so long as these differences remain. Thus in summer the current will follow (Fig. 99) the arrows *S*; in the fall little or no current will be set up; in the winter the current will reverse and follow the arrows *W*; in the spring the conditions are again nearly balanced, and little current will flow. When the shaft attains a depth of 800 feet, the subterranean air is always hotter and lighter than the surface air at any season; and unless the two outlets have a great difference in elevation, an uninterrupted current will continue, without fear of reversal, down the lower and shorter opening.

While this method may be universally practised under favorable conditions in metal-mines, it is evident that in collieries one danger arises from the reversal of current, for at one time the current, following the arrows marked *S*, carries the air through the gangways, whence it is distributed among the work-rooms, to be returned to the surface by way of the longer and deeper shaft; but during the other season the air may follow the arrows marked *W*, thus entering the working places first, and departing thence through the gangways, makes its exit by the lower or shorter shaft. If, now, there be a number of abandoned rooms or goaves connected with the working rooms, it is manifest that in the latter season the air must pass through them first before reaching the men at work, and thus carry noxious gases with the current to spread calamity by explosion or fire. Again, the fact that no air will circulate during the vernal seasons would render the provision for supplementary means of ventilation imperative. Air-currents which have served for ventilating preparatory workings or prospecting drifts in the virgin seam never should pass over stalls or working places where men are engaged, on its way to the air-level.

45. The atmospheric air which surrounds us possesses, in

common with all other gases, in consequence of the **repulsion** between its molecules, a tendency to expand into a **greater** space. This indefinite expansion, by reason of which every gaseous fluid, not restricted by an extraneous force, continues to expand to the tenuity of interstellar space, results in the creation of an air-current whenever by an increase of temperature or a diminution of pressure the given mass of air expands in opposition to the attraction of the earth and rises into the upper strata. This upward flow will continue so long as the gas expands until the resistance encountered by it is equal to, or greater than, the repulsion among its molecules. It is this readiness with which gases tend to adjust themselves to the varying conditions of temperature and pressure that plays so important a part in mine-ventilation. The tension of a gas increases with the condensation, and the density of a given mass of air is proportional to its tension; or, since the spaces occupied by one and the same mass are inversely proportional to their densities, the volumes u and u' occupied by it are inversely proportional to its relative densities p and p' . The energy stored up by a given quantity of air, when compressed to a certain degree, may be measured by the work restored by it in expanding, and this energy may be converted into motion producing a current, or it may result in a pressure when that tendency to motion is resisted, or when the motion is suddenly arrested.

The volumes, u , assumed by a given weight of a gas are inversely as the corresponding pressures per unit of surface :

$$u : u' :: p' : p.$$

If the temperatures change while the pressures are constant, the volumes, reduced to absolute zero (-461° F.), will be found to vary proportionally.

$$u : u' :: 461 + t : 461 + T;$$

t and T being, respectively, for up and $u'p'$, Fahrenheit readings.

The weight of a cubic foot of air at a temperature t , and a barometric pressure B , in inches of mercury, is obtained by the following formula, and at a temperature T , is W , expressed as follows :

$$w = \frac{1.3253B}{461 + t}, \quad W = \frac{1.3253B}{461 + T}.$$

That portion of the energy stored up in the air which is expended during its expansion in dynamic effect causes a "wind" or "draught," the velocity of which depends upon the difference in tension.

The velocity with which gaseous particles will move, whether their temperature has been increased or their pressure decreased, is measured by the formula

$$v = \sqrt{2gH};$$

in which H is the head due to the difference between the tensions, or densities, of the initial state of the condensed gas and the final state of the expanded gas. Atmospheric air, which, normally, is under a barometric pressure of 29.92 inches and a temperature of 32° F., when flowing into a vacuum attains a velocity, in feet per second, which is equal to

$$\begin{aligned} v &= \sqrt{2gH} = 8.02 \sqrt{\frac{P}{W}} \\ &= 8.02 \sqrt{\frac{14.7 \times 144}{0.00118}} = 10758.9 \text{ feet.} \end{aligned}$$

The total difference of pressures per square foot is represented by P in pounds, and the weight of a cubic foot of the warmer or attenuated gas by W .

So, too, the velocity with which compressed air or steam escaping freely from a pipe or other reservoir of the same may be ascertained, the value to be supplied for H , the head to which the velocity will be due, being equal to the pressure in pounds per square inch under which the gas exists, multiplied by 144 and divided by the weight in pounds per cubic foot of the exhausted fluid.

When, however, two masses of air of equal height but of different tensions, p and p' , are exerting a pressure upon one another through a connecting conduit, the resulting difference in pressure per unit of area of base measures the motive force; in which case P is the total difference of aerostatic pressure in pounds per horizontal square foot of sectional area of base, and W the weight per cubic foot of the rising column of air.

If, then, a column of air at t° F., D feet high, with a base of one square foot,

be heated to T° F., its new height would be greater by some quantity which we may call M . If two such columns be connected, being of the same depth but of different temperatures t and T respectively, the latter column would be lighter than that at t° by a quantity $D(w - W)$; and so long as this difference in temperature is maintained, this difference of pressure, which we may represent by P , ensues, by reason of which the hot column of air would be driven upward, producing a draught with a velocity, V , due to the aerostatic head M . To hold this force, P , in equilibrium would require a resistance $D(w - W)$ pounds per square foot; or the pressure of an additional column of warm air weighing W pounds per cubic foot, of a height of M feet,

$$M = \frac{D(w - W)}{W} = \frac{P}{W} = D \frac{T - t}{461 + t}.$$

This quantity M is known as the motive column to which is due the velocity of the flow of air, and if no resistance is offered to it, motion will take place. It may be represented by OT (Fig. 99), which equalizes the pressure of the unequally heated columns of air below the level of the line OO .

When two such shafts are of unequal depth, as at O and M in Fig. 99, and have equal exterior and interior temperatures, a rarefaction of the air in either one of them not affecting the other would result in a diminished pressure upon the bottom, just as is obtained by a difference in temperatures; a rising current is established therein, with a velocity dependent upon the ratio $\frac{P}{W}$, in which P is the difference in the weights of the two shaft columns of a base one horizontal square foot in area and a height D ; and W is the weight of a cubic foot of the rarefied air.

For the purpose of mine-ventilation there will be required a motive column much larger than that here obtained, because of the enormous friction of the air in rubbing along the rough surface of the workings, turning sharp corners, and squeezing through small openings. The resistance due to this cause amounts often to as much as 90 per cent of the power. In other words, only one tenth of the theoretical motive column becomes effective in producing a current, and the actual velocity of the air-current, v , does not exceed one third of the theoretical velocity, V , due to the head M .

The principle upon which chimney draughts for boiler or other heating apparatus depends is also similar to that here

described, excluding, of course, frictional allowance. In chimneys for boiler-furnace draughts, the fire burns best when W is $0.5w$, and the height of the manometric column in the chimney is about one half an inch of water.

It is evident, therefore, that the height of a motive column depends upon the difference in temperatures or a difference in tensions, or both, of the gaseous mixture contained in the two shafts or entries to the mine. A measure for this motive column may be had in feet of head of pressure per square foot of area of the base, or in the number of inches of a water column in the manometer which corresponds to this weight. Insomuch as a column of water one inch high and a square foot in area of base weighs 5.184 pounds, the height, m , of a water-gauge column which will balance the pressure P is equal to $P \div 5.184$.

Let M be the head corresponding to the motive column, v the velocity of flow of the upcast air per second; then is $\frac{v^2}{2g}$ the effective velocity-head of the issuing air; and if W is the weight of a cubic foot of the warm or attenuated rising air, the theoretical energy of the moving air per second is WM , and the effective or actual energy is $W\frac{v^2}{2g}$, or $0.01553Wv^2$.

That portion of the energy which is consumed in overcoming the friction of the mine is therefore $W(M - 0.01553v^2)$. It is this lost energy which is measured by the water-gauge. As the mine resistances are reduced, so the water-gauge reading is reduced, and the efficiency of the mine increases, permitting a greater actual return from the expenditure of the same potential force.

The height, m , of the water column being measured in inches, the number of horse-powers, H , necessary to produce a ventilating current of Q cubic feet per minute is known by the following formula:

$$H = 0.0001571Qm.$$

The indication, therefore, which the water-gauge reading

gives of the ventilating force is evident in the above formula—that for a given quantity of air, Q , in circulation, the horsepower necessary to produce ventilation increases with the resistance of the mine.

46. There are several methods by which the natural ventilation may be accelerated and properly distributed to meet all the requirements of the mine, each of which contemplates some method either of decreasing the tension of the mine air to enable the return current to ascend to the surface, or of increasing its tension by the use of a compressor to force atmospheric air into the mine. The several means by which these results are attained may be, first, a furnace built at the bottom of the outlet shaft, or a fan; or, second, a blowing, propelling, or air-compressing fan at the mouth of the inlet shaft. By either of these methods a different state of tension is produced in the two shafts connected below, and, in the effort to establish equilibrium, the air is set in motion, a draft is created, and a current is established that flows through the air courses at a velocity dependent upon the head due to the difference in pressures, as has been seen in **45**.

Furnaces are employed for increasing the temperature and are constructed in such manner as to be remote from direct contact with the coal, yet in close proximity to the shaft which constitutes the outlet for the mine air, and in a gallery through which circulates air from the workings. The pit selected for the outlet should be that one which would naturally carry the flow in winter. The furnace is simply a fireplace, walled and roofed by a fire-brick or common-brick arch (Figs. 100 and 101). When special care is taken, a second wall is built outside and over, with an air-space between, to isolate it from the coal and prevent fire. If the roof is wet, a double arch must surmount the furnace, as otherwise the steam generated will burst the arch. If the mine is fiery, or considerable dust is floating, care must be taken that the gases are well diffused, or else the current must not be brought into close proximity with the fire. In such cases the current is split, a small portion being heated over the fire, the remainder passing through

a "dumb-channel," entering the upcast 50 feet or so above. A still safer plan passes all of the fiery current through the

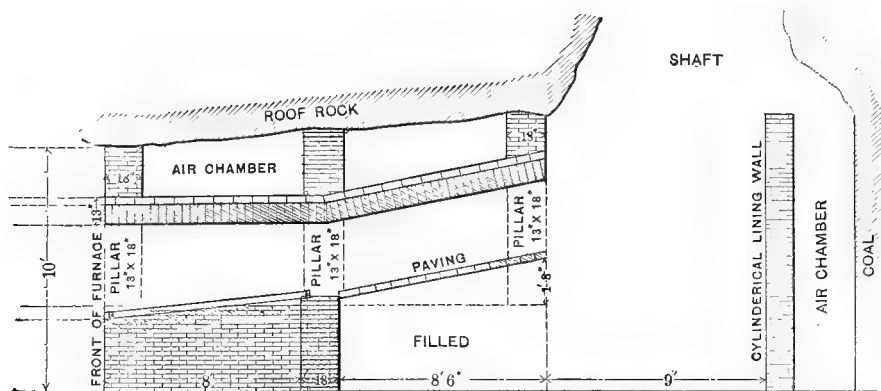


FIG. 100

channel, and feeds the furnace by a split current of fresh air direct from the intake.

The size of the grate depends upon the work to be done. Its bars are 3 feet from the floor, slanting upward toward the

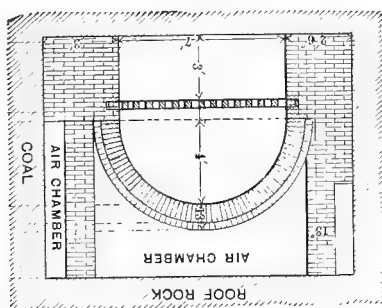


FIG. 101.

shaft 1 to 6, distance to the roof 4 or 5 feet. The width wall to wall, is 6 feet and its length from 4 to 12 feet, according to the volume of air to be moved, which is about 1500

cu. ft. per square foot of fire surface on a properly constructed furnace.

An ordinary furnace of 34 sq. ft. heating-surface, costing \$130, will heat a column of air such as will furnish 29,000 cu. ft. per minute. A large number of furnaces 10×12 furnish 200,000 cu. ft. The cross-sectional area must be 50 per cent greater than the upcast air-way, and the shape capable of regulation by double sliding iron doors, to produce varying degrees of contraction and of combustion. The fire is spread over its entire width, and over only as much of its length as is necessary to furnish an adequate motive column, at a temperature of 140° to 160° F. Emergencies, as low barometer and high thermometer, and the cleaning of the grates, require other and more heating-surface. The coal consumed is 2 to 5 tons per day, spread thin and evenly over the bars, and fed from both ends, on a long furnace. This rate is 40 to 70 lbs. per hourly h. p. of work done on the air. Attendance, etc., is \$5 per day.

Q being the quantity of air in cubic feet per minute, W the weight of a cubic foot of return air, T being the temperature, $F.^{\circ}$, of the upcast air, and t that of the air in the return air-way, the number of pounds of coal consumed by the furnace per hour is

$$x = 0.00376WQ(T - t).$$

The area, F , of the grate-surface in square feet is about one tenth of the hourly coal consumption, in pounds, and its relation to the depth, D , of the furnace below the surface is known by the expression

$$FQP = 1,716,000 \sqrt{D},$$

P being the manometric depression in pounds per square foot. and Q , the volume of air per minute.

An arrangement which is so simple and so cheap in construction besides being easy of management presents advantages which have long commended it to mine operators; nevertheless the difficulties with its use, the dangers which

attend the exposing of an open fire in gaseous districts without the possibility of introducing a safeguard, the numerous calamities traced to the furnace which has fired either the solid coal surrounding it, the gases in the return-air, the timbers of the shaft, or even the surface plant, and its lack of economy in shallow pits, were soon made manifest. The atmospheric changes of the seasons reduce its efficiency, a decrease in the barometric pressure and an increase in the surface atmosphere reduced the action of the furnace, and notwithstanding its great superiority over many other mechanical appliances it has gradually been supplanted by fans. The power of the furnace increases arithmetically with the temperature, and that with the amount of fuel burned. The quantity of coal that can be consumed upon a given area is limited. The resistance of the mine (see 50) increases on the other hand geometrically with the square of the velocity of the current, and it is therefore manifest that between the several conditions the furnace limit is soon reached. Many furnaces may be cited supplying to the mine over 200,000 cubic feet of air per minute; and enormous as they are, their cost is very little less than that of a modern fan of large size; but when we contemplate the huge pile of coal thus consumed for the production of the current, we are forced to the conclusion that efficient furnace ventilation is a luxury which the coal trade cannot long endure. Perhaps as the depth of the collieries increases to about 2000 feet that the furnace may be reinstated.

With the atmospheric air at 62° F. and the furnace-heated air at 132° F., the water-gauge depressions, *m*, produced at various depths of furnace are as follows:

<i>D.</i>	<i>m.</i>	<i>D.</i>	<i>m.</i>
50 feet,	0.086 inch.	1000 feet	1.735 inch.
400 "	0.694 "	2000 "	3.471 "
700 "	1.215 "	4000 "	6.943 "

These are in accordance with the formula for estimating the manometric depression:

$$m = D \frac{W}{5.184} \left(\frac{T - t}{461 + T} \right).$$

Ex. 20.—A colliery has two shafts 1000 ft. deep, 12 ft. in diameter; temperature in the downcast is 60° F.; barometric pressure is 30 ins. 150,000 cu. ft. of air are supplied per minute by a furnace. Required the temperature of the upcast and the horse-power necessary to produce the ventilation, the mine being supposed to show a water-gauge resistance of 2 inches of water.

520° F., and 163.5 h.-p.

Assume the coefficient of friction for the smooth shafts to be as great as that of the rough mine galleries; then

$$p = \frac{0.0000000217 \times 1000 \times 37.7 \times (150,000)^2}{(113)^3} = 12.8.$$

Each shaft therefore offers a resistance equal to 12.8 lbs. per square foot. The total resistance then is $25.6 + 10.368 = 35.968$. The work done in ventilating is $150,000 \times 35.968 = 5,395,500$ ft.-lbs., or 163.5 horse-power.

The temperature of the upcast shaft is

$$\frac{P}{W} = \frac{(461 + T)35.968}{39.759} = 1000 \frac{T - 60}{521}.$$

Or, by another method: A cubic foot of air at 60° F. and 30° barometer weighs 0.0766 lb. 150,000 cu. ft. of the circulating air weigh 11,490 lbs. Since the furnace is performing 5,395,500 ft.-lbs. of work upon 11,490 lbs. of air, the height through which it is moved is 470 ft. in one minute. Then M is 470 ft., $t = 60^\circ$ F., and $D = 1000$ ft.

$$T = \frac{M(461 + t)}{D - M} + t = 520^\circ \text{ F.}$$

From this it is seen that the temperature of the upcast air necessary to force 150,000 cu. ft. of air through the mine is dangerously high. The furnace must be replaced by the exhaust-fan, or the frictional resistances must be reduced by enlarging the entry-ways.

What should be the size of the air-way shafts in the above case, that the upcast air be not hotter than 190° F.? By substitution above, M is found to be 200 ft.; the work is then $11,490 \times 200 = 2,300,000$ ft.-lbs. (70 h. p.). This value requires that p should not exceed 15.3 lbs., which limits the shaft's resistances to 4.932 lbs., or 2.466 lbs. each. In order to obtain so low a friction, the areas are enlarged to a radius of 16.66 ft.

$$pa^3 = flmq^2, \text{ or } p(\pi r^2)^3 = fl(2\pi r)q^2.$$

47. Mechanical ventilators include a variety of devices, of which fans remain our main reliance at the present time. As the furnace has in the past supplanted various mechanical devices in the form of pumps and trompes, so fans built on various principles have succeeded the furnace and the steam-jet. There are two classes or types of fans: (1) blowers, either rotary or reciprocating, and (2) fans, propeller or centrifugal.

Those of one type sweep out a fixed volume of air at each revolution and are known as the definite-volume exhausters, under which head come the Root, Baker, Lemielle, Cooke, and Fabry. In the other class, acting centrifugally upon the air, we have a simple revolving wheel always working in one direction, producing by its rotation a pressure or a rarefaction the degree of which depends upon its speed. Of these we have the Guibal, Waddle, Walker, and Schiele ventilators.

The trompe is a simple application of the injector principle,—water falling in the cylinder and carrying with it air, creates a small intake draught. The volume of air, compared with the quantity of water used, is so insignificant that, unless an especially favorable means be provided for carrying off the water, the ventilation is too expensive to be continued except as a temporary expedient.

The blowers, either rotary or reciprocating in their action are of general use in America, being represented by the Root, Baker, and Champion on the one hand, and air-compressor and other reciprocators on the other. The blower forces the air through the intake compartment of the mine, which discharges it at the upcast. These blowers or force-fans are much in vogue for small workings and as expedients in furnishing a separate ventilation for stopes and drifts; but few are employed in coal-mines to produce the total ventilation there required. In metal-mines, however, they are largely depended upon, though they supply a pressure higher than that ordinarily required to overcome the resistance of the mine. They produce air by reason of their high speed at a pressure often attaining ten pounds per square inch, whereas a mine requires an initial pressure only sufficient to overcome its resistance, which is rarely greater than ten pounds per square foot of base. The blower is a small radial wheel revolving freely in a casing and nearly touching its sides. By a central opening on either side the air is admitted to be acted on and set into rotary motion. These blowers may be had in sizes capable of furnishing as much as 16,000 cubic feet per minute, requiring from one to fifteen horse-power for their

operation. Some of the blowers are capable of a ready alteration from a blower to an exhaust, or the reverse, which fact recommends them particularly for wide shafts which are liable to freeze during winter. This is particularly advantageous in metal-mines, where it makes very little difference which way the current moves. In collieries, however, as has been seen, this is not feasible.

The Root blower or force-fan consists of two interlocking impellers revolving side by side in very close connection, without actually touching one another or the enclosing case. They are made of cast iron accurately bored and dressed to a true surface, so that, while practically no air escapes, there is also no internal wear. At each revolution a definite volume of air enters, is enclosed, and discharged either at the top, the bottom, or the side. They are driven by a pair of external gears, at a speed ordinarily of from 250 to 500 revolutions per minute. The extremities of the revolving arms of the impeller section are of an acorn shape, or their surfaces are arcs either of true circles or of cycloids.

The Fabry, which resembles the Root blower, is much used in the north of France and Belgium. Two fans, each having three broad blades arranged radially, are hung in a chamber. They revolve with equal velocities in opposite directions, the blades coming in contact, isolating a quantity of air, and expelling it into the atmosphere. The success of this blower is attributed to the fact that there are no joints in it.

The Baker rotary force-fan has inside of its casing three drums, each being an independent casting turning truly and balanced perfectly to insure a steady motion. The upper drum, which receives the power from the engine, does all the work of blowing, while the two lower drums serve as valves to prevent the air from escaping.

Cooke's is a positive machine. An eccentric drum revolves inside of a 12-foot circular case very close to which is held a swinging shutter that cuts off the entering—from the discharge—current. The inlet and outlet portion occupies 235° of a revolution. At Lofthouse iron-mines are seen two of these

side by side, the drums being placed opposite each other on the shaft, so that the revolving mass is balanced, the discharge equalized, and the efficiency raised.

The Lemielle is a species of rotary air-pump, complicated and leaky, producing large volumes under great rarefaction. It consists of a vertical cylinder, within which a second revolves eccentrically; on this latter are two or more vanes, which in one part of the revolution lie close to the shutter, and in another open and expel the air.

The reciprocating blowers have been displaced almost entirely by the rotary blowers, either class being capable of a reversal of rotation to force air into or exhaust air from the mine, as desired. The power required to drive the force-fan depends upon the volume and pressure of air exhausted or discharged; but the rule usually followed for computing the net power in a given volume at different pressures is to multiply the number of cubic feet delivered per minute by the pressure in pounds per square foot at the blower, and the product by 0.00003; the quotient will give the net horsepower required to drive the fan.

The centrifugal fans, which are used almost exclusively in this country, may be divided into two great classes: (1) those which are called open-running, by which we mean that they are free and discharge their air all around the circumference; and (2) those called close-running fans, which have but a restricted opening for the discharge of the air. Those of either class are made large in diameter and are driven at a relatively small angular velocity, though few, such as the Schiele, are of small diameter, running at a high angular velocity. They produce large volumes of air at a low pressure, and may be reversed in motion to exhaust or to force air. The diameter of the fans of this class may be and is occasionally as high as 50 feet, those of small diameter being regarded as unnecessarily cumbrous. The action of all fans is based upon the general law that bodies in motion tend to travel in straight lines, resisting any attempt at diversion from this path, in consequence of which, when the fan is set in

motion, its blades come in contact with its interior air, the particles of which are at rest and resist rotation. When, however, the particles do move, their endeavor to travel in straight lines results in their making for the circumference, producing thereby in the central portion of the fan a partial vacuum, which is replaced by the air external to the fan. So long as the rotation of the blades continues, so long will this current be produced and maintained, the pressure of which will increase as the peripheral speed increases.

Fans which are of the open-running variety include the Waddle, Biram, Naysmith, and Hopton, all of which are essentially similar to the first named. The Waddle is a self-contained fan in that there is no fixed casing, and the whole machine revolves. Its form is practically that of a light hollow disk of wrought iron, the blades and casing being wholly riveted together. The air enters by a straight lead at one side only, and passes through curved and gradually narrowing channels to the circumference, the blades being bent at first to incline slightly backwards, the alternate blades extending not more than one half the distance between the circumference and the inlet. The passages, by their contraction, are so made that the circumference at any point multiplied by the cross-sectional area at that point is a constant quantity. The outer circumference of the fan is bell-mouthed.

A fan of 9 feet diameter circulates 80,000 cubic feet with a water-gauge of 2 inches. One of 45 feet, driven by an engine with 40" $\times$ 42" cylinder at a boiler-pressure of 80 lbs per square inch, has given a volume of over 550,000 cubic feet at 42 revolutions.

The Hopton has an inlet on each side of the central diaphragm with backward-curving blades, and a construction very simple. The revolving portion consists of the arms and blades working between two brick walls.

The open-running fans must, in order to be efficient, discharge their air at a very low velocity, because the energy of bodies in motion increases as the square of the velocity, and

that passed by the discharged air is, therefore, so much useless work. It is for this reason that the passages in the more correct open-running fans, like that of the Waddle, are curved backward. The theoretical depression which can be produced in fans of this type is equal to the height due to its peripheral speed, T , in feet per second.

$$H = T^2 \div 2g = 0.01553 T^2.$$

The closed-running fans are essentially of a more massive structure than those of the open-running type, being of considerable width as well as of diameter. Of this class of fans the Guibal is a type, the Scheile and the Walker Indestructible being similar in construction. Inside of a fire-proof housing a horizontal shaft is revolved by an engine or dynamo, carrying with it an hexagonal or square frame, on which are built six or eight blades. The blades are flat and slightly curved at their tips, sometimes radially and often inclined backwards. The clearance between the tips of the blades and the casing is made as little as possible, except for a certain distance at the bottom, through which the air is discharged, the amount of that opening being regulated by an adjustable shutter in a gradually enlarging chimney. The air enters at the centre, whence it passes into one of the intervals between the consecutive blades, which form an *évasée* canal, the speed of exit being less than the speed of entry (Fig. 102).

In the Scheile fan the blades are contracted in width from inlet to outlet, the fan being surrounded by the usual spiral casing, into which the air discharges all around the circumference, the space continually increasing until it reaches the chimney. The blades of the Rateau fan extend to the centre of the fan, and have a peculiar curvature slightly forward, and also a curvature in the line of the fan-shaft. Immediately in front of them is a cone terminating in a point. The Capell fan, of equal power with the Guibal, is smaller, and runs at a higher speed. It has two concentric shells besides its outer casing, in each of which are curved blades with the convex side

forward. The air enters the inner shell, is forced out through ports into the second outer shell, where it strikes the concave face of the outer blade, and thence is discharged at a low velocity through the usual expanding exhaust-flue.

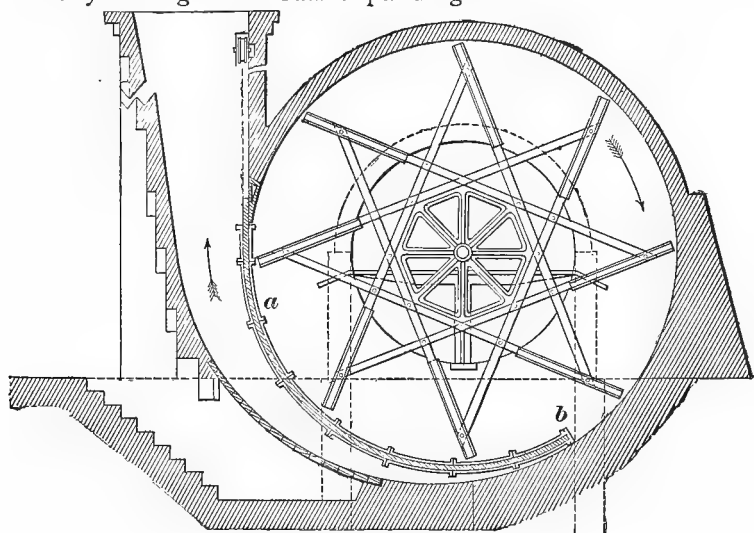
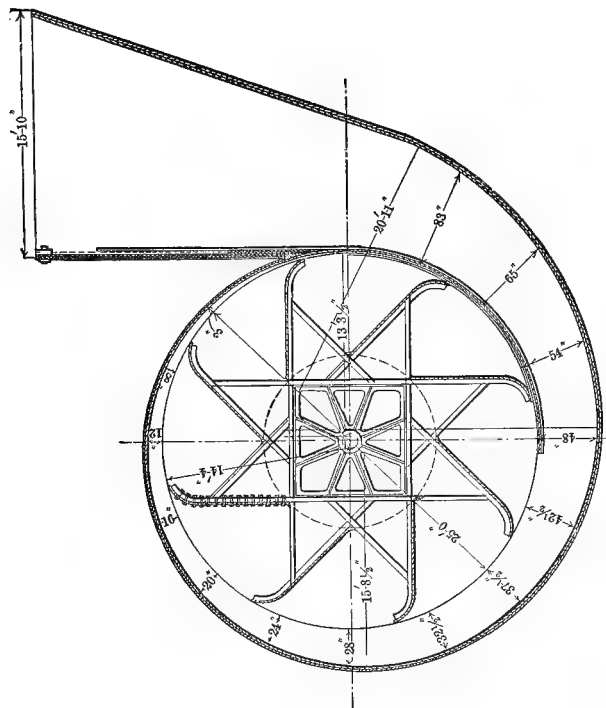


FIG. 102.—FIGURES OF WORKING DRAWINGS OF FAN.

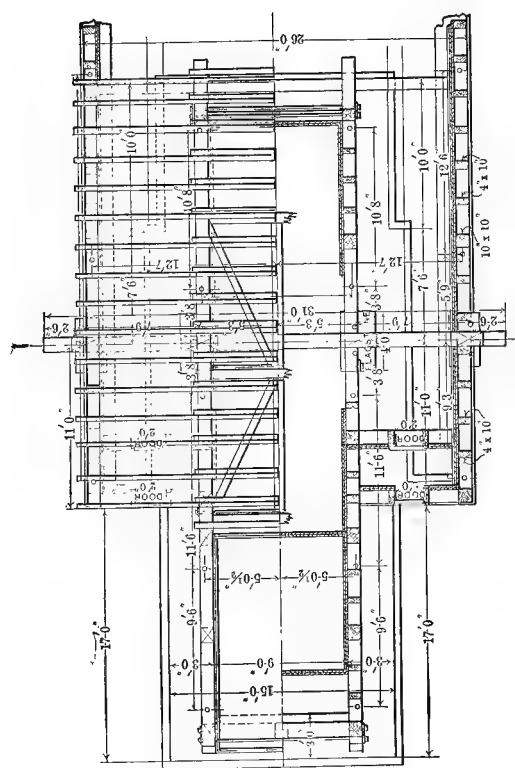
The special improvement giving rise to the name of “anti-vibration shutter” is made after the manner of an inverted elongated V, and constitutes the characteristic of the Walker fan, in which the injurious rebound produced with every revolution of the blade in the similar types of fan is remedied, thus enabling a higher speed with less wear and tear, and a practically silent fan.

The Champion fan, which is really two fans joined together by a common centre ring, is designed to propel the air with a minimum resistance, the blades having a backward curvature. The use of the inner casing or hood and attendant diaphragm, which are hung on frames, renders it possible to change the current at will, blowing to exhaust, by revolving the hood around the fan without stopping the latter.

The theoretical depression produced by a covered venti-



SECTION OF FAN CASING.



SECTION ON LINE A-B
PLAN

FIG. 104.

lator with an expanding chimney is twice that of the uncovered or open-running type, and is equal to double the height due to the tangential speed of its blade-tips. The use of the chimney gives to this type of fan the enormous advantage over the other that the air may be discharged from the fan at a higher velocity without any material loss of energy. The gradually increasing space into which they discharge reduces the velocity and utilizes all the energy in giving motion to the air, while the air is ultimately sent out into the open at such a speed that no resistance practically is experienced. The fans of either type are of dimensions as large as 50 feet in diameter and 12 feet in width, those of the open-running class being comparatively very narrow. The volumes which these fans will produce vary directly as the speed of their rotation, and their depression varies as the square of the speed of rotation. Though they may be run at any speed at will, the efficiency of the fan materially decreases when the speed of the tips of the blades exceeds, to a great degree, 5000 feet per minute, or is less than this quantity. The rate, however, which is regarded as normal is 4000 feet of peripheral velocity per minute. Below or above the normal speed a loss of velocity ensues in the discharging air, which alternately is expelled into the chimney or carried with the blades into the fan, there to repeat its circuit. The discharge is frequently followed by a vibration in the fan, to remedy which the sliding shutter (*ab*, Fig. 86) is introduced. Its use permits of a high speed and efficiency; and its correct position is only known by experiment in each individual case, to determine by the point at which the throbbing ceases with the given speed. Numerous experiments have been conducted upon centrifugal ventilators with the view of determining the influence which the various dimensions of the fan and shapes of its parts will have upon its performance; and the following conclusions are cited from the results of the tests made by R. Van A. Norris, Wilkesbarre, Pa., upon 25 fans, as the influence of: "1st. The diameter on their performance seems *nil*; the only advantage of large fans being in greater

width and a lower speed required of the engines. 2d. Width upon efficiency is, as a rule, small. 3d. Shape of blades shows that the back curvature is better, and diminishes the vibration. 4th. Shape of casing is considerable. The proper shape would be one of such form that the air between each pair of blades would constantly and freely discharge into the space between the fan and casing, the whole being swept to the *évasée* chimney. A large spiral, beginning at or near the point of cut-off, gives in every case a large efficiency. 5th. The shutter on the fan is beneficial. The exit area can be regulated to suit the varying quantity of air, and prevent re-entries. 6th. Speed at which the fan is run. The efficiency is high if the peripheral velocity is large."

In many states the law requires all ventilators to be provided with a recording instrument by which the number of revolutions of the fan shall be registered every hour and such data to be taken and reported. In other states also is required an automatic regulator for the water-gauge. The speed-registers are generally constructed of a metal pedestal erected on blocks at the side of the fan or engine-shaft, a small vertical shaft to which a governor is attached. A small cog-wheel on the lower end geared to a large driver on the fan or engine-shaft communicates the speed to the governor, which, by a system of leverage, raises or lowers the arm to which is attached a pen that presses against a paper dial held in position by a light case of sheet brass. The higher the speed of the fan, the more will the governor raise the lever, and consequently the pen register. The time is recorded by a clock to whose shaft the dial case is attached. In other devices the dial case is a cylinder in which is rolled a sheet of paper turning on a horizontal axis, which is also the continuation of the shaft of the clock. These instruments perform the work expected of them with great satisfaction.

The ventilation of a mine by a fan is affected by atmospheric changes in a manner similar to furnace-currents, a low barometer or a high temperature requiring an increased de-

gree of rarefaction from the fan or furnace. Moreover, as the depth of the mine increases the work devolving upon the fan proportionally increases, because normally the air becomes denser; with every additional thousand feet of depth, an increased rarefaction or depression of 0.4 inch of water-gauge is necessary. Compared with the furnace its efficiency decreases with the depth of the upcast until, at a certain depth, it becomes an open question between the relative merits and demerits of fan and furnace, as to which will be the more economical. For shallow works, the exhaust-fan undoubtedly takes precedence. At the depth of a thousand feet a large furnace will equal a very imperfect fan, consuming 20 pounds of fuel per hourly horse-power; a good fan and condensing engine will be cheaper than a furnace down to the depth of about 4000 feet. Taking cognizance of the objections to the furnace, it must also be borne in mind that machine ventilators are subject to serious objections, since during the time of their repair the mine must remain unventilated, whereas with a furnace after its fire has been extinguished a considerable circulation will still continue in the upcast for some time. Auxiliary ventilating appliances should be supplied against any emergency which arises during the repair of the fan.

48. The theoretical depression of a fan is the height of a motive column of the density of the flowing air and is equal to the ventilating pressure exerted by the fan, friction being left out of question, whether of the mine or of the fan and its mechanism. An ideal ventilator will produce a depression which is twice the height created by the tangential speed of the tips of the blades. If, then, H be the height of the motive column due to the velocity, T , of the fan tips in feet per second, H will equal $\frac{T^2}{g}$; but imperfections of detail prevent such an initial depression being attained, and representing them by a coefficient K , which is always less than unity, —reaching 0.85 in Guibal fans, but more often being below 0.6 in the average construction of fans,—the fan approaches

an ideally perfect one when K approximates to unity. The yield of the fan then in barometric depression, or its useful effect, is $H = \frac{KT^2}{g}$. Various enfeebling causes modify the capacity of the fan to determine the value for K . The quantity of air which passes through an orifice is equal to the product of the area and the velocity when no friction is encountered; but when any fluid flows through an orifice in a plane surface a considerable diminution of the discharge takes place, because the directions of molecular flow converge and produce a contraction of stream. The coefficient corresponding to this contraction, known as the *vena contracta*, is 0.65; hence with a given velocity, T , and a head, H , under the conditions modified by the coefficient K as above, the discharge of air per second will become

$$q = 0.9194a \sqrt{KT^2}.$$

Hence it is evident that if the capacity of the mine is such that it is incapable of delivering to the fan the volume of air equal to the body capacity of the latter at a given speed, the frictional resistances encountered in the mine will reduce the efficiency of the fan by some quantity which is usually comprehended in the symbol a , representing the area of the mine's "equivalent orifice" in square feet. Experiments have demonstrated that when a is 20 square feet, only 65,000 cubic feet of air are obtained per minute for the fan peripheral speed of 5000 feet per minute; but when the mine resistances have been reduced until its "equivalent orifice" is as large as 100 square feet, 280,000 cubic feet of air are obtained from the same speed of fan. The value of this fiction, which in earlier days was known as the temperament of the mine, enables us to grasp the conditions under which the ventilator is working.

In like manner the equivalent orifice of the fan, which is designated by the symbol o , may be determined. It measures or represents the orifice in a thin plate which offers such a resistance to the flow of the current, Q , as is equal in effect

to the aggregate resistances encountered within the fan from its imperfections. If H is the theoretical depression which the fan should produce when moving at a tangential speed, T , per second, and h represents the actual or the effective depression which is produced upon the air as measured by the water-gauge, then $H - h$ is the head wasted by the fan in its construction and may be represented by h_0 , which measures the head corresponding to the equivalent orifice. In large fans its value varies from 16 to 80 square feet.

The head lost in the fan, represented by h_0 , is equal to $H - h$, the velocity due to which may be determined by the expression

$$v_0 = \sqrt{2gh_0}.$$

As the value of h_0 approaches zero and that of h approaches H , the fan approaches an ideally perfect ventilator. The actual velocity through the orifice of entry is $0.65v_0$, whence the area of the orifice o , which equals the quantity flowing per second, divided by the velocity of the flow, has the following value:

$$o = \frac{q}{0.65 \sqrt{2gh_0}} = \sqrt{\frac{q^2}{27.87h_0}}.$$

The density of water being 833 times that of air, the ratio between the water-gauge reading and the height of the motive column, H , is 1 : 833. To convert the water-gauge reading to a height H of air-column of equal weight in feet, the height of the water-gauge, m , in inches is multiplied by 69.4.

The ratio between the lost head in the ventilator and the effective head represented by the water-gauge is expressed in the equation

$$\frac{h_0}{h} = \frac{a^2}{o^2}, \quad \text{and} \quad h = H - h \frac{a^2}{o^2},$$

whence

$$h = \frac{o^2 H}{o^2 + a^2}.$$

The quantity of flow through the mine and also through a fan, depending on the relations which the area of the mine air-ways and the condition of their rubbing surface bear to the mechanical condition of the fan, is manifestly dependent upon a proper ratio of a to ϕ , which ratio may be expressed as the "appropriateness of the fan to the mine." When this value is equal to or greater than unity, the fan would be too small for the mine, and it is questionable whether any air would flow under those conditions. As the ratio becomes smaller, the conditions become more favorable for the fan. When approximating a ratio of 0.3 the orifice of discharge of the fan is to be considered as having a fair working ratio. More air is obtained by a given fan and at a given velocity when a is large than when a is small, for, no matter how well constructed the fan may be, it cannot provide a quantity equal to its body capacity unless the mine can pass this quantity. The effective work done upon the air is less in the latter case than in the former for a given volume of air. The mechanical work of centrifugal force is $0.0000340(T^2 - V_2^2)$. In this T is the circumferential velocity and V_2 is the absolute velocity at expulsion, due to compression from centrifugal force. As V_2 increases, so the work on the departing air, and proportionately the effective work, decreases. The use of the funnel-chamber reduces this quantity to $\frac{1}{4}$ or $\frac{1}{6}$, and the work lost to 4 or 5 per cent.

The efficiency of the fan is measured by the ratio between the actual centrifugal pressure, F , and the effective pressure, p . The mechanical efficiency is also measured by dividing the horse-power in the air by the engine-duty. With fans properly constructed, the efficiency approximates about 68 per cent. In experimentally measuring the efficiency of a fan, it is customary to determine the dynamometric resistance and internal friction when its orifices have been cut off from any communication with the mine, the air being then drawn from the atmosphere and, after passing through a fan, expelled at its throat. Counting the rate of revolution and estimating the volume of air which is moved, the power necessary to

overcome this friction is determined and expressed in feet of the air-column whose weight equals the aggregate friction. This quantity divided by the theoretical head corresponding to the velocity of the fan determines its efficiency under the conditions named. The fan is giving its maximum efficiency when "its body capacity just exceeds the quantity the mine will pass at a gauge pressure, F , due to the speed of rotation of the fan."

49. In designing a fan to give an ample service to the mine, the essential elements are Q and m . These given, the diameter, the peripheral speed, and the length and width of blades, as well as the direction of their inclination, must be determined by the engineer. As to diameter, it may be said that the slow-running fans are regarded as cumbersome and costly, requiring expensive foundations. Large fans may be run at a lower rate of revolution and produce the same tangential speed than would a fan of small diameter. Insomuch as speed is the important factor in the construction of ventilators, due consideration must be given to this question, which is determined by local conditions of place, economy, and mechanical simplicity. A convenient rate of revolution for a fan directly connected with the engine is about 60 per minute. The body capacity of the fan should be large enough to maintain the required pressure, P , without great variations in speed. Though the practice of European engineers tends toward the rate of tangential speed which represents 5000 feet per minute and over, in this country 4000 may be considered as normal. In any event, if the calculation and design be made on the assumption of either normal speed, it will be possible, when an emergency arises, to increase the speed sufficiently to give a volume nearly one tenth greater than the normal quantity. Moreover, when the mine is dry and dusty it will be possible to turn the whole volume of the excess into any or each single split, through which it may be drawn, clearing away fine dust and moisture.

The entry for the air should be made easy and large,

preferably divided into two inlets, one on each side, with a diaphragm to prevent the currents from conflicting. This necessitates a wide fan, which, however, gives a volume proportionately greater than what is to be had from a single fan with a single large inlet.

The length of the blades of the fan should be only a little greater radially than the difference between the radii of the fan and its inlet. With a large inlet the blade necessarily is shortened, and when pressure is desired the blade length should be increased to as large a quantity as possible by providing two inlets. Notwithstanding that the width of fans is much greater than would be obtained by substitution in the formulæ following, it is certain that the latter dimensions correspond to a greater efficiency. M. G. Hanarte concludes that "the Guibal fan has always been eight or nine times too wide, and the Capell is nearly as bad."

The shape of the blade should be such as would present to the circumference of the outlet an inclination following the resultant of the movement of rotation and of the movement of the air penetrating the spaces between them. The blades of open-running fans curve backward. The backward curvature is conceded to give a freer delivery, and the forward curvature at the tips a higher water-gauge pressure. The number of blades is seemingly a matter of indifference, though the limit may doubtless be determined by the inevitable friction produced by the excessive surface of contact when too numerous. The friction varies as the cube of the section of space between two vanes. As to the shape to be given to the casing, it will be noticed that the original Guibal fan had no spiral, the tips of the blades revolving but two inches clear of the casing, and the spiral enlargement beginning at the angle of about $67^{\circ} 30'$ from the lower vertical radius. Those fans presenting a large spiral beginning at or near the cut-off and increasing about six inches for each 45° up to 275° , and thence widening by an increasing increment to the *évasée* chimney, appear to give larger efficiency by allowing for the slackening of the speed of the air, and discharge the air with less energy at the exit.

M. G. Hanarte concludes that the spiral envelope is not necessary.

Below are given formulæ for the computation of dimensions of a Guibal fan in accordance with the data indicated above. All dimensions are in feet.

D = diameter of the fan between the blade-tips : $Q \div 200 = D^2$;

l = length of the blades in feet ;

r = their radial length = $2.61m$;

x = their width = $A : 2\pi s$;

A = aggregate area of the one or two inlet-ports in square feet (radius of each central inlet, s) = $Q \div 1300$;

N = number of revolutions per minute ;

T = peripheral speed of fan per second = $DN \div 19.0985$;

V = theoretical velocity per second due to head H ;

v_1 = velocity of the centre of gyration of air-column between the blades
= $\pi(D - r)N \div 60$;

ρ = radius of gyration of the mass of air = $\frac{1}{2}(D - r)$.

W = weight of the unit of revolving air-column per foot of fan width
= $0.0766r$;

F = centrifugal force of the air-mass in pounds per square foot of discharge area or of the housing = $Wv_1^2 \div \pi g = 0.0007578rv_1^2$;

V_2 = velocity of air discharge per second = $\sqrt{\frac{(F - P) 1,800,000}{2130 + P^2}}$;

Z = minimum area of discharge-port = $A \div 2$;

h_0 = fan resistance, measured in feet of head, = $H - h$;

O = area of orifice offering a resistance to the flow of Q cubic feet of air per minute, equal to that of the fan ;

q = quantity of air discharged by the fan per second in cubic feet = $0.65v_2z$;

Q = quantity of air discharged by the fan per minute in cubic feet = $60q$;

m = mine resistance in inches of water-gauge ;

P = mine resistance in pounds per square foot = $5.184 m$.

In Figs. 103 to 106 are illustrated the details of the ordinary pattern of fan which is designed in accordance with the conditions indicated above. As, fortunately, neither the Guibal fan nor the shutter is subject to patent, the working drawings here given may aid the construction engineer.

When the conditions are satisfied by the revolution of the fan of proper proportions, the centrifugal pressure of the fan should produce a depression, F , equal to or exceeding P , the mine resistance, in order that the requisite discharge through the outlet should equal the desired quantity Q . When it is discovered that the volume of discharge is deficient, the fan

dimensions D or r should be enlarged or the rate of revolution increased. Below is a brief table indicating the theoretical water-gauge depression in inches for the corresponding peripheral speeds in feet per second.

T	m	T	m
49.2135	1.084	82.023	3.008
56.103	1.408	88.588	3.510
63.321	1.794	94.818	3.967
66.930	2.004	100.078	4.481
76.117	2.593	106.301	5.056

EXAMPLE.—Required the dimensions of a fan to provide 125,000 cubic feet of air against a mine resistance of 2.5 inches.

At a normal rate of 65 feet per second, the diameter becomes 25 feet; the area of the inlets is 96 square feet, the diameter being 11 feet; the radial length of the blade is 6.5 feet; the minimum width of the blade is to be 2.8 feet.

As $\rho = 9.25$ feet and $v_1 = 51$ feet, the centrifugal pressure, F , becomes 12.7 pounds per square foot of radial column; and the velocity of discharge 31.6 feet per second, which with a minimum area of discharge-port, Z , of 48 square feet, would furnish less than 60,000 cubic feet. The mine resistance exceeds the standard allowance of one inch of water-gauge for each one hundred thousand cubic feet of air. The mine air-ways should be enlarged or the fan operated at a higher speed. An increased rate of 70 revolutions per minute will produce a ventilating pressure of 22 pounds per square foot. The blades may be lengthened and two inlet orifices be provided, each of 48 square feet in area.

At the peripheral velocity, T , of 91.66 feet per second the theoretical head of discharge is 261 feet. But the effective head, h , against which the fan is operating, measured by the water-gauge, is 166.66 feet. Under the conditions of operation, therefore, the loss of head, h_0 , in the fan is 94.34 feet; since the equivalent orifice of the mine is 21.2 square feet, the equivalent orifice of the fan, O , is 41.2 feet. The ratio of a to O , nearly one half, represents a fair working ratio of appropriateness of fan to mine.

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CHAPTER XIII.

DISTRIBUTION OF THE AIR.

50. Calculation of the work done in ventilating a mine; losses by friction; coefficient of friction; formulæ; examples; similarity between the formulæ for frictional resistances of water, air, and electricity; examples and illustrations. 51. Interpretation of water-gauge readings; formulæ; examples; Buddle's system of splitting air-currents; advantages and economy of the plan; principles of dividing air-currents into panels; formulæ; laws governing the area of airways; dangers of goaves, and the necessity for their isolation. 52. Velocity of the air and the modes of measuring it, by candle, smoke, or anemometer; place for observation; calculation of the ventilating power. References.

50. It has been assumed thus far that the work done upon the air is totally effective in the mine; that with a given M and P the calculated quantity of air is obtained without any loss; that the momentum, once imparted to the air, would carry it through the mine and out. This is not so. Friction instantly overcomes the momentum; the velocities given by the formulæ (Lecture 47) are never realized in practice. The rough sides of the galleries and rooms, their sharp corners, and the diminished areas offer resistances to the passage of the current that consume often 90 per cent of the power. Moreover, the subtle air under pressure seeks to escape at every opportunity, and some portion of the precious fluid is lost into the goaf, through doors and at crossings. A certain mine theoretically required a pressure of but 1.2 lbs. per foot to give rise to its current, yet the friction was such that 11.8 lbs. were actually necessary to create the velocity. Not infrequently the ratio between M (to which the generation of the final velocity at the top is due), and M' , the head actually necessary to overcome resistances, is as low as 1:18. In other words, only 5.5 per

cent of the work done upon the air is usefully expended. Any means of reducing this loss is to be welcomed.

Let us examine into the laws governing the movement of fluids and their applications to the conditions, that we may learn to reduce this friction to a minimum, and obtain salubrity, safety, and economy with the least outlay. The air which enters the mine from the downcast is distributed to the rooms and chambers in proportions varying with their several needs; or the current as one mass sweeps through the main way, along working faces, thence by return air-way over the furnace or to the fan. The resistances encountered depend upon the ratio of the area of the surface rubbed to the area of the conduit, and upon the coefficient of air-friction against rock. A satisfactory value for the coefficient has not been obtained: the records of experiments show it to vary as in water, according to the nature of the conduit and the velocity of the flow. The coefficient varies with the nature of the rubbing-surface, and consequently differs in various air-passages of the same mine; nevertheless, the numerous experimenters have announced values for the coefficient of friction of air in mines for each foot of rubbing-surface and for a velocity of one foot per minute as varying between 0.00000008585 and 0.000000219, with the preference given to the latter quantity. This value for f , the coefficient, is measured in the pressure per square foot in decimals of a pound. Measuring the height of an air-column in decimals of a foot, the value for the coefficient of friction f' has been found to be 0.00000010635 to 0.00000026881, the two extremes of values for f' being heads of air-column corresponding to the values given for dynamic pressures, f . The latter values, in both cases determined by J. J. Atkinson, are most frequently used, and though higher than any of the values ascertained by other experimenters, err on the side of safety, and hence are accepted as the constant coefficient.

Let l be the length, m the perimeter, and a the area of the gangway, through which the air is coursing at v feet per minute, and the rubbing friction is found experimentally to be $flmv^2$. Imagine a piston, fitting air tight in the passage;

to just move it against the resistance requires the expenditure of a force ϕa , in units of lbs. and sq. ft. Therefore the loss of power due to friction is $\phi a = flmv^2$, and the loss of head in feet, $\phi = \frac{flmv^2}{a}$, or in lbs., $\phi' = \frac{flmQ^2}{a^3}$.

This cannot be ignored, for, other things being equal, the quantity of air received at any face is inversely as the resistances encountered on the way. In the "splitting" of the air it is of special import. It will be observed that the frictional loss is directly as the perimeter and inversely as the area. This would suggest the desirability of selecting such a shape for the air-way as will make it as spacious as circumstances will permit, consistent with a diminution of the exposed surface. The circular form most nearly meets this requirement; but, as a rule, we are restricted to the rectangular or the more advantageous trapezoidal cross-sections. Two galleries 5×5 require one third more power to carry the same amount of air as a $5' \times 10'$ gallery. It will also be noted that in galleries of equal cross-section the volume of air passing through them having the same resistance will be inversely as the square roots of their lengths. The gallery which is 1600 feet long and carries 6000 cu. ft., offers the same resistance and consumes the same amount of power as one of equal area 711 feet long, delivering 9000 cu. ft.

The friction increases with the square of the velocity. So it would be far better, desiring a given quantity per minute ($Q = va$), to increase the area rather than the velocity. Conversely, a local contraction of the air-passage, by the use of a partly opened door, a pile of waste or of gangue, will materially diminish the air passing through it.

A comparison of the above formula for air with those for electricity and water will show an identity of loss, though in different units; for electricity it is $P = C^2R = fLC^2 \div a$; in which C represents the quantity of electricity flowing through the wire; and for water Kutter's formula for determining the resistance to its flow in pipes is $a = clmv^2$.

51. It has already been remarked (44) that the water-gauge measures the drag of the air in the mine, and thus serves to

indicate the pressure and head corresponding to the motive column M . The pressure varies from $\frac{3}{4}''$ for easy to $4''$ for difficult ventilation (from 3.9 to 20.7 lbs. per sq. ft.). In anthracite mines it is about $2''$. The motive column, which is to just maintain this pressure against resistances, should also be sufficient to create a final or exit velocity in the shaft. If the entire current traverses the mine unbroken, the resistance in the shaft or entry is only a fractional part of the mine friction indicated by the water-gauge, and the following formulæ apply with sufficient accuracy:

$$V^2 = \frac{pa}{flm}, \quad Q^2 = \frac{a^3 p}{flm};$$

The value for f is to be taken always in the same terms as that for p . In other words, if the mine resistance, p , be given in pounds per square foot, the corresponding value for f is taken as equal to 0.0000000219; or if the value for p is given in feet of head of motive column, M , the value for f is then 0.000000269.

If the air-ways in the mine, the resistances of which are to be calculated with a view to determining the necessary ventilator pressure to produce circulation, are all of the same dimensions, the calculation of the lost pressure may be made in one operation by proper substitution for the length, periphery, and area of air-way and the velocity or quantity concerned. The value of the frictional resistance, p , thus engendered in the mine corresponding to the water-gauge height, m , and of the velocity of the air-current, added to that of the pressure, p'' , requisite for the generation of the velocity, determines the motor pressure required. Often p'' is very small compared to p , and may be even neglected without sensible error; but when it is large the actual ventilating pressure, which must be supplied by the force-fan, or the manometric depression to be produced by a furnace, or exhaust fan, must be such as exceeds the sum of $p + p''$.

When the air-ways of the mine are of various cross-sections, the resistance offered by them in the aggregate must

be determined by adding together the separate values for p , calculated for each differing cross-sectional area and length. When the air-current is "split" into several smaller branches, and circulated through an equal number of divisions of the mine, more or less equal in length, with volumes of greater or less velocity, the value for p must be calculated in each division or district separately; and for each differing air-way the aggregate resistance in each division is the sum of the resistances encountered in each of its various galleries. The sum of the several frictional losses of head or of pressure, and that pressure or head which produces the final velocity at the mouth of the mine, is again equal to the ventilating pressure demanded of the motor.

Formerly it was the practice to meander the air through all the galleries of each lift before expelling it (Fig. 10). This involved heavy pressures, enormous air-ways, or a velocity dangerously fast, and the last gang, fed by the departing current, would receive an irrespirable atmosphere, vitiated by the emanations from all previous sources. There was nothing to commend this pernicious system, and it is certainly a matter of congratulation that it is becoming obsolete.

Many years ago Mr. J. Buddle introduced a system of ventilation for fiery mines that has everything in its favor. This system was known at first as "coursing the air," and now is termed "splitting the air," the inception of which is due to Carlisle Spedding or his son of Whitehaven who introduced it in 1763. By it the aggregate quantity of air is increased, the dangers of explosion are lessened by confining its train of evils to one portion of the mine, and power for ventilation and haulage is saved, since it goes hand in hand with the method of panel-working (Fig. 12). Each panel of the mine is completely isolated from the contiguous districts by barrier pillars, and is ventilated separately by delivery to it of a portion of the volume of the intake which does service in that panel, to be afterwards discharged into the return air-way, where it rejoins the exhaust from the other districts. The electric distribution for purposes of illumination and the water-

supply of a town are conducted on identically the same principle, i.e., that which recognizes the tendency of a fluid to seek a shorter and easier escape from confinement. With a number of conduits receiving at a common point a volume of fluid from a larger conductor, each will convey a fractional amount of that original bulk which is inversely proportional to the resistance offered by its entire rubbing-surface. If the several conduits again meet to discharge their individual volumes of fluid at a common point into a common reservoir, the pressure at the point of discharge is the same at the mouth of each and every pipe. Likewise the pressure at the point of union is the same in the ends of each and every pipe. The loss of head or of pressure due to the flow of the given quantity of fluid through each conduit is then the same. If the original bulk is allowed naturally to subdivide, the amount of fluid in the several branches will vary in an inverse ratio with the cross-sectional area of their conduits. This is equally true of the circulation of air through mine galleries or districts, of water through branching pipes, or of electricity through connecting wires in the circuit. In planning, therefore, the ventilating system for the mine which is divided into a number of districts for ventilation purposes, the practice is to calculate for each separate district its aggregate resistance to the flow of the volume required for a known number of men employed there, and for a dilution of the gases evolved in that district. Several separate values for p are thus obtained. But these district resistances must be equalized or else the inlet-current will be so subdivided at the point of distribution that the large bulk of the air will pass through that district which offers the least resistance; while to that district offering the greater resistance the volume there circulating will be small. This is usually the reverse of the requirements; for, generally, that district offering the smaller resistance is the shorter one, having less men in its circuit, and therefore requiring a smaller volume of air; while that district presenting the greater resistance to the flow of the current is either more extensive, has a greater volume of

goaves, or contains more working places, and hence demands a very large fractional part of the main current. In order, then, to automatically deliver to that district requiring more air, which at the same time offers a greater resistance, the area of the conduit throughout its course or the area of the orifice at the central point of distribution must be made sufficiently large as to tempt through it, or into it, the requisite amount of air, leaving to the smaller district, which requires less air, an orifice of entry which is comparatively small. By so doing the differences in pressure for each and every district, between the point at which the splits of the fresh-air current are made and the point at which the return-currents from the same splits reunite, can be equalized to that of the one offering the greatest resistance. Then the current will naturally divide according to the areas of the inlets or of the passages, and each district will receive its apportioned fraction of the incoming air. Hence, whenever the ventilation of the mine is to be split into several currents and the air is to be apportioned in accordance with the demand, the mine foreman, having calculated the relative values for the head lost in each, determines by proportion, as will be seen, the comparative area of inlet to be provided the several districts at the point of distribution, and there, by means of doors and other regulators, does so furnish to the given district the area desired.

The measurements of the water-gauge pressure or loss in head between the beginning and the end of the split and the velocity of the flow of air, are made in the intake; and while it is not always possible to subdivide the current at a common point of distribution, this should be done as near to the down-cast as circumstances will admit. The same may be said of the point of reunion. Otherwise the resistance of the intermediate ways and of the entries must be determined and provided for, as may be seen in the example given below. The aggregate resistance of the intermediate ways of the several districts through which the air is circulated determines the maximum number of splits which are possible.

Simple as is the theory, and satisfactory and economical

as is the plan when well developed, it is not easy of execution. The success of the plan involves an exact manipulation and great skill in taking due precautions to balance the various factors, to determine the equilibrium designed, and to prevent one panel or district from receiving too brisk a current at the expense of others. Hence, while it is eminently desirable to apply this theoretical distribution, its difficulty is recognized, and it has become the practice of the foreman to approximate the desired conditions by making repeated tests upon the quantity of air flowing in a given circuit, which, if insufficient, is provided for by enlarging the inlet area for the given district and watching its reaction upon the volumes in the other dependent splits. In shallow workings, though the mine may be extensive, the practice is an inexact one in many cases. It may often be cheaper to sink a new shaft to furnish separate ventilation to a district, than it would be to undertake to furnish an elaborate system of splits.

Though it may not require a demonstration to show that the subdivided splits of the current are productive of greater economy in ventilating power, attention will be called to the fact that the ventilating force in h.p., necessary to deliver a volume, Q , against a mine resistance p , is measured by the expression

$$\text{h.p.} = Qp \div 33000 = Qm \div 6365.7.$$

The resistance which would be offered by the aggregate of all the districts to the flow of the entire volume, Q , through the whole length of the circuit is measured by p . Each fractional volume, q , q' , q'' , etc., passing through only one branch of the circuit would offer a resistance r , r' , r'' , etc., which is very small compared with p . When the mine boss has adjusted the regulator doors at the point of distribution by altering the respective areas of inlets, the resistances in all of the several circuits are equalized, the work performed in each split is qr , $q'r$, $q''r$, etc., and the aggregate ventilating power is their sum. As

$$q + q' + q'' + \dots = Q \quad \text{and} \quad r < p,$$

the power required for the ventilation in branches will be less than that for a single current Q , through the same passages.

The power required for 16,200 cu. ft. of air flowing in one column would be capable of producing 70,884 cu. ft. of air in five splits, 94,850 in ten splits, and nearly 100,000 cu. ft. in fifteen splits.

Ex. 21.—A colliery is ventilated by a Guibal fan of 21' 3" diameter, making 40 revolutions per minute. How many cubic feet will it produce? The air must pass through a main air way 300 feet long, 6×12 feet, before being split up into three separate air-ways, one being 12,000 feet long, 6×5 feet; another 11,000 feet, of area 6×7 feet; while the third is 10,000 feet long and 5×8 feet in section. Required also the water-gauge pressure, assuming the two shafts together to consume 0.226 lb. per square foot in friction.

16,380 cu. ft. and 0.9 inch.

The splits are all drawn from a common point of junction.

Theoretically, the fan produces a water-gauge pressure of 0.902 inch (4.677 lbs.). Then the entire mine offers a resistance of 3.504 lbs. (0.676 inch). The resistance of the main air-way is $p = 0.000000006279Q^2$. Q , the quantity of air delivered, is divided up into three several sections, according to their resistances. As p is the same for each, the quantities q may be known in terms of p to be

$$2171 \sqrt{p}, \quad 3455 \sqrt{p}, \quad \text{and} \quad 3368 \sqrt{p}.$$

Now the resistance of the entire mine is equal to 3.504 lbs., plus that of the main air-way, plus that of the splits, p . From this we know $p = 3.504 - 0.000000006279(8994 \sqrt{p})^2$, whence p becomes 3.333 and the resistance of the main air-way 0.170 lb. Q then becomes 16,380 cubic feet, and the quantities received by the three splits are 3940, 6270, and 6110 cubic feet. (The difference in results arises from failing to carry out the decimals beyond two places.)

Ex. 22.—An air-way 3000 feet long, 8×4 feet area, is carrying 20,000 cubic feet. How many feet would be produced if the air was split into three currents, the power remaining the same? The sections are 3000 feet long and 8×4 feet area; 3600 feet and 5×9 ; and 4800 ft. of 6×10 feet.

51,736 cu. ft. and 11.56 h.p.

The calculated power necessary to drive the quantity of air, Q , through the three sections is equal to the sum of the three powers, par , of each section.

The benefits that may be derived from splitting the air-current are manifest by inspection of the following case:

Ex. 23.—A mine has two slope entries, 9×14 feet in cross-section and 100 feet long, and such internal resistances as would be equivalent to 8000 feet of a typical air-way (see page 446) 5×10 feet in cross-section. What pressure and power will be requisite to propel 16,200 cubic feet?

$$p = 0.2619 \text{ and } u = 4243 \text{ ft.-lbs., for the two entries, and} \\ p = 11.19 \text{ and } u = 181,383 \text{ for the total.}$$

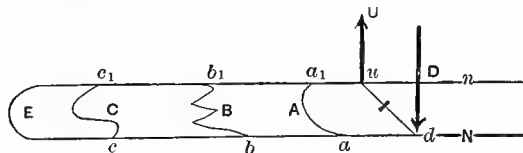
Ex. 24.—Required the quantities of air that will circulate where there are 2, 3, 5, 10, and 15 equal splits, the pressure remaining the same as above.

After calculating the pressure p for the one current as above, then proceed to ascertain the pressure p' necessary to circulate 16,200 cu. ft. in the several cases. These will be found 1.369, 0.405, 0.0874, 0.0109, and 0.0032 lb. per square foot, respectively; the areas, be it remembered, for the equal splits (see page 447) are 100, 150, 250, 500, and 750 sq. ft. in the several cases, while the rubbing surfaces, lm , are the same (240,000 sq. ft.). The pressures are then apportioned directly to 10,935 lbs., the mine friction of one current as above. Whence, the pressures being as the squares of the volumes circulating, we obtain 33,409 cu. ft., 66,354 cu. ft., 91,692 cu. ft., 103,755 cu. ft., and 105,255 cu. ft., and 373,887 ft.-lbs., 742,504 ft.-lbs., 1,026,030 ft.-lbs., 1,161,010 ft.-lbs., and 1,177,760 ft.-lbs. as the respective powers u .

Ex. 25.—If it be desired to know what quantities will circulate with the same power u , as in Ex. 29, then we have but to apportion the volumes to the cube-roots of the powers, u , of Ex. 30.

Thus $\sqrt[3]{373,887} : 33,409 :: \sqrt[3]{181,383} : 26,252$, the volume with 2 splits,
 $\sqrt[3]{742,504} : 66,354 :: \sqrt[3]{181,383} : 41,479$, “ “ “ 3 “
 $\sqrt[3]{1,026,030} : 91,692 :: \sqrt[3]{181,383} : 51,461$, “ “ “ 5 “
 $\sqrt[3]{1,161,010} : 103,755 :: \sqrt[3]{181,383} : 55,881$, “ “ “ 10 “
 $\sqrt[3]{1,177,760} : 105,255 :: \sqrt[3]{181,383} : 56,419$, “ “ “ 15 “

Ex. 26.—When, however, the splits are not taken from a common point of juncture, the procedure for ascertaining the mine resistances, and, subsequently, for balancing the delivery of air to the several sections, is not so simple. As explained on page 203, the plan consists in determining the several resistances and the powers necessary to overcome them. These are then added as follows: Fig. 244 illustrates a case. D is the downcast shaft, 846 feet deep,



8 × 10 ft. in cross-section, delivering 56,000 cu. ft. per minute; by force-fan, 27,750 cu. feet go to the left gangway, while the right gangway passes 28,250 cu. ft. of air. The water-gauge stands at $\frac{3}{4}$ inch (4.525 lbs.). Required the volumes of air received by the splits A , B , C , and D .

The distances along the gangway at which the splits are taken are a , 460 feet from d ; b , 960; and c , 1360 feet from d . Dimensions of gangway 6 × 12 feet.

The splits are E , receiving 5000 cubic feet through 100 feet of 6 × 12 ft. gangway; 60 ft. of 4 × 2 break-through; and 60 feet of return air-way 5 × 14

feet; *C*, having a resistance equivalent to 2401 feet of typical roadway 15×6.87 feet, connecting with the return-way at c_1 , 941 feet from the upcast *U*; *B*, which has 220 feet of gateway 4×7 ft., 1400 ft. of entry 4×8 ft., and 1000 ft. of room 4×12 , delivering at b_1 a point 581 ft. from *U*; and *A*, which has a resistance equivalent to 1810 feet of typical roadway 5×10 ft., connecting with the return air-way 313 feet from *U*.

The upcast shaft is 870 feet deep and 12 ft. in diameter. It was found that the velocity of the outgoing air was nearly 708 feet. This would correspond to about 80,000 cu. ft. of air.

In the return air-way, along ua_1 , the volume was found to be 41,610 feet. This shows an increase in volume of 13,870 cu. ft., which may be accounted for by the higher temperature of the outtake air or increments to the original volume from gas.

Beginning at *E*, where 5000 cu. ft. is circulating, we find the total resistance to be such as requires a pressure of 0.096 lb. per sq. ft. at c_1 to overcome it. Hence the same pressure must exist at the mouth of *C*, where $pa = 0.00000021967q^2$. Equating the two values for p (q being unknown for *C*, and 5000 cu. ft. for *E*), we find that the volume of air of 6410 cu. ft. offers a proportionate resistance to that of *E* (see page 204).

At the point b the total resistances of *E*, *C*, and their gangways must be proportional to those of *B*, as the relative volumes to be circulated. At b the 11,410 cu. ft. passing out produce a pressure of 0.57 lb. per sq. ft., at which rate, by equating this with the sum of the resistances of the split *B*, we find that q becomes 3720 cu. ft.

In like manner 12,580 cu. ft. will be found capable of passing through *A*, the frictional resistance at a being 1.19 lbs. per sq. ft.

The total resistances pa of the various sections are: da , 53.33 lbs.; *A*, 74.59 lbs.; ab , 17.29 lbs.; *B*, 16.04 lbs.; bc , 7.83 lbs.; *C*, 9 lbs.; *E*, 6.92 lbs.; c_1b_1 , 17.67 lbs.; b_1a_1 , 23.29 lbs.; a_1u , 91.2 lbs.; *D*, 323.85 lbs.; and *U*, 356.2 lbs.

The volumes of air are, respectively, 27,740 cu. ft.; 12,580 cu. ft.; 15,150 cu. ft.; 3720 cu. ft.; 11,410 cu. ft.; 5910 cu. ft.; 5000 cu. ft.; 17,115 cu. ft.; 22,725 cu. ft.; 41,610 cu. ft.; 80,000 cu. ft.; and 56,000 cu. ft.

The pressure mentioned is only that necessary to overcome the rubbing friction of the moving air against the sides. No cognizance has been taken of the pressure requisite to air the mine. That power will depend upon the velocity, and is equal to about 0.17 lb. pressure per foot.

The power requisite to force the air into and out of the mine (11.9 lbs. per sq. ft.) is 666,400 ft.-lbs.

Ex. 27.—An air-way 5×8 ft. across, 6000 ft. long, is followed by a through 2×5 ft. in cross-section 300 ft. long. To drive 10,000 cu. ft. of air through them requires 5.5 h. p. Their frictional resistances are equivalent to 5.2893 lbs. and 9.114 lbs. per sq. ft., respectively; the total resistances, pa , being 211.57 and 91.14 lbs. A pressure of 302.71 lbs. is therefore requisite at the inlet to overcome frictions of the passage. $302.71 \div 40 \times 10,000 = 75,667$ ft.-lbs. to propel the air through the first section, in addition to which it requires $9.114 \times 10,000$ lbs. to overcome the resistance of the second.

When the "through" precedes the airway, we have no difference in frictional resistances, but the total pressure in the first section must be such as to overcome the total resistance of the two, hence it is 302.71 lbs. (30.27 lbs. per sq. ft.). Then $30.27 \times 10,000 + 5.289 \times 10,000 = 355.601 \text{ ft.-lbs.} = 10.7 \text{ h. p.}$

If the "through" is midway along the 6000 ft. air-way and has 300 ft. of the $5 \times 8 \text{ ft.}$ roadway on each side, we still have the same resistances, yet the power requisite for propulsion is 299,047 ft.-lbs. For each end the friction is 2.645 lbs. per sq. ft., and for the middle it is still 9.114 lbs. per sq. ft. The total pressure of the first is 105.784 lbs.; of the second, 91.14 lbs.; and of the last, 105.784 lbs. The pressure at the inlet is 302.71 lbs. (7.5677 lbs. per sq. ft.); at the entrance to the second, 196.924 lbs. (19.6924 lbs. per sq. ft.); and at the entrance to the last, 105.784 lbs. (2.645 lbs. per sq. ft.).

An inspection of the several examples included above will demonstrate the necessity for spacious air-drifts, since they do not merely reduce the resistance to the flow of the current through them, but, as will be seen later, they reduce the velocity of the flow to a degree which is comfortable for miners walking against the current, and to a degree of safety for lamps which must be impermeable to an atmosphere of gas at a specified limited speed.

Goaves are the most dangerous places. It is estimated that the air-space in a goaf is one sixth of the volume of coal extracted, and in it most likely will breed a great deal of gas, of which the sweating of the roof is an infallible sign. So also is the exudation of sulphuretted hydrogen. Often a water-gauge placed in a goaf stopping will indicate by the difference of level in its arms whether or not any accumulation of gas exists behind the stopping. Spontaneous combustion once begun therein, nothing will stop it. For this reason, though aëration is possible, fiery coals should only be worked by a method involving complete removal of the coal, or its replacement by clean waste.

52. To maintain a distribution of the air through the workings requires a thorough organization and rigid supervision. While the salubrity of the air is preserved by an immediate removal of the mephitic gases, a velocity exceeding 500 feet per minute is not desirable; it is neither comfortable nor safe. A speed exceeding 500 feet is equally injurious with stagnation. Several mining commissions of Europe have experimentally determined that, aside from the chilling effect of walking against

so rapid and cool a breeze, lamps are not safe: a rapid current incites explosion by driving the gases through, or the flame against, the screens. Many lamps can resist higher-speed currents, but none are safe in over 900 feet per minute. In the old country the air-currents at different parts of the mines vary in velocity—at the coal face often as fast as 900 feet; but here our Davy, Stephenson, or Clanny lamps require protection in such a velocity, which exceeds that of American practice.

A miner can approximately estimate the speed of the current by knowing the rate at which he must walk to keep the flame erect, or by noting the time elapsing between the discharge of a volatile fluid or smoke and the time of its arrival at a point a known distance beyond. The anemometer is, however, much the simpler instrument for measuring the velocity of the flowing air. In a case is a series of vanes which are moved by the current, and these by proper gearing turn indices over their respective dials at such rate that the velocity may be at once read. It does not give accurate

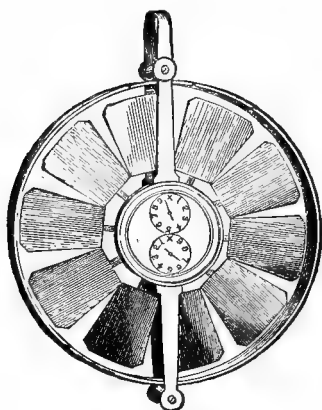


FIG. 106.

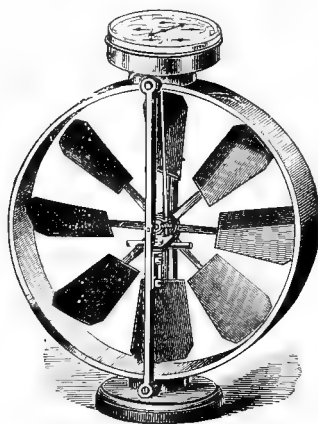


FIG. 107.

results on account of the friction of its mechanism. Each instrument has its own factor, which is not even constant. Biram's and Costello's patterns (Figs. 106 and 107) are most used in America. Their factors are ascertained by occasional

test, the anemometer being revolved on a whirling table, and its reading compared with the actual velocity of revolution.

The point selected for observing the velocity should be in a straight gallery, whose sides and roof are a fair average in roughness, and where there is neither a sudden bulge nor a contraction. The average of several one-minute readings are taken at the place of measurement, near the roof, sides, and centrally. Then the cross-section of the conduit at the observing station is taken. The product is the volume circulating.

The power U to move the air is vaP , which, divided by 33,000 ft.-lbs., gives the actual h. p.; and the ratio between these observations, simultaneous with the indicator-diagrams of the engine, gives the efficiency. The ultimate comparison is with the coal consumed, which approximates 11 lbs. per useful h. p. by fan, and 40 to 70 by furnace. In designing a motor, a good margin should be granted for emergencies.

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CHAPTER XIV.

REGULATION OF THE AIR-CURRENT.

53. Doors, regulators, etc.; safety doors, and extras, to be dropped after explosion; air-crossings, overcasts, brattices, and their use; mineralized brattice. 54. Complete example for the ventilation of a mine, with two outlets and five splits; furnace, fan, and natural ventilation methods compared; example and calculation for a railroad tunnel. References.

53. THE air which enters the mine at the lowest point possible is conducted to the men by a route as direct as the plan of the workings admits. To those who are at work in the shaft and at the breast of the exploration entries, the current is led by a partition in shaft or drift forming conduits for an intake and return current. To the men engaged in extracting coal, the current passes from the main entry to the heading, thence through cross-headings, or branch headings, to the rooms. The air-current passing down the entries by either shaft, or slope, is split at each level or lift to a right and a left branch. In each lift the current flows to the furthestmost point, along the main heading, thence rising in to the most remote room, along its face, thence through "dog-holes," progressing toward the hoistway till each room has been traversed, when the current is led to the return or back heading which communicates to the upcast way, there to be joined by the return current from the opposite side of the lift, and finally to unite with the return current rising from the lower lifts which have been fed in like manner. The main heading is usually the haulage-way, the back heading serving only for the vitiated air, though it is also used for the travelling way. The jaw of each room, except that one which

is most advanced, is stopped up, which latter, in time, is closed as another room is turned from the heading. As each room must be in free communication with the haulage-way for the delivery of its coal, the stopping there provided must be capable of opening for the passage of the coal. The swinging doors, therefore, which are placed in the entry are always closed to prevent the air from returning at once down the back heading. Here, doors are built to deflect it into the rooms, with also a brattice in each room, to direct the current it receives from its neighbor. From the last room, whether working or abandoned, the air passes to the return current. In the lower portions of the room not in the direct sweep of the air-current the air is prevented from becoming stagnant by the leakage from the main current and the addition of such volume as is swept in or out by the travel of the cars from and to the haulage-way. The distribution in the long-wall is perfectly similar, in which case the gob roads are stopped up by doors to convey the current to the extremities of the workings, whence the air flows along the long-wall face. In flat seams, the double-entries of which have a low grade for haulage from the surface, the similar split system is employed, the splits being taken at the point of union with the main headings. When, by reason of the low pitch, three entries are driven from daylight, the common practice consists in making the central entry the return air-way, the air being led into the mine in two currents on either side. This simplifies the subdivision of the currents, which may be effected with fewer doors; but each district on either side will have a crossing of its return current with the intake current. Each set of rooms feeds to one and the same haulage-way, constituting one district for ventilating purposes.

This review of the systems of ventilation will prove the classification of the means for deflecting the current as of seven varieties: (1) swinging doors in gob roads and at jaws of room to prevent the entry of air thence from the main heading; (2) swinging doors in the return headings at appropriate places to prevent the immediate return of the current

and to deflect it to a district or room,—these latter are removed when the working advances 100 feet or so beyond its present place, to be again set up with the advance; (3) stoppings or wooden walls closing the “throughs” when their service as connectors has been performed, the other connection in the same room 100 feet above having been effected; (4) partitions of wood or canvas, to temporarily divide each air-way into two compartments for an inlet to the face, and for the outgo to the cross-heading nearest to the face connected with the return air-way, back-heading, or nearest room; (5) brattices of canvas hung from the roof in a common travelling way, to serve as light temporary resistances to a current flow of low pressure without impeding the access to the rooms for the men; (6) air-bridges constructed to transmit one current, usually the return, over or under one of two air-ways which intersect; (7) sliding doors placed in the branch air-ways, and so fixed as to offer an area for the inlets to the several branches, including the continuation of the main way. This inlet is usually the haulage-way, though frequently a difficulty arises when the area of the regulator doors is small.

In the rooms, entries, and shafts the distance between the working place and the last connection with the air-way must not exceed 22 yards, and that from the end of the brattice must not exceed one fifth of that distance.

“Throughs” or “dog-holes” are driven at every 30 yards through the pillar coal to connect rooms, and to allow of air circulation through them in turn as the room advances.

For intensifying the permanent air-current led to a working place, or for the separate ventilation of workings in seams with slight disengagement of gas, compressed air may be employed or hand-worked fans may be used for separate ventilation, which is always brought up so near to the working place that the air be not too much diffused.

The doors are of two classes—those which may be provided with an additional valve or gate, which may be opened or closed within a limited range; and those doors swinging on

hinges, which may be opened for purposes of ingress or egress; they should open against the direction of a possible inlet current, in order to completely direct the current along the main way. Doors are the main dependence for the ventilation of mines, particularly those which are subdivided into districts for purposes of ventilation. They are placed in such position as will temporarily and effectually check and deflect the current. Wherever their location, they must be built with great care, of matched timber, closely fitted in their frame, and are maintained only so long as the draught is to be kept up. Some are even provided with weather-strips on their edges. They are placed in all the side entries to the rooms, swinging outward towards the main passageway, and are generally in pairs when located in the haulage-way. In the latter case they are located far enough apart that the two will not be open at the same time, and thus interrupt the principal circulation.

Regulator-doors are also employed at the mouth of entry to a ventilating district in the mine, the gate of which constitutes an adjustable sliding door capable of being secured against disturbances, and are employed to regulate the supply of air to be delivered to that district in accordance with the system described in 53. The placing of the slides is left to the mine-boss or the fire-boss. Those doors which are placed in haulage-ways either require an attendant or have their frames so inclined that the door swings to of itself. Automatic mechanical appliances for opening and closing the doors from a distance without stopping the mules or other haulage-motors are employed in many collieries, but do not stand in great favor. While there seems to be no means yet offered for replacing doors or dispensing with them entirely, there is no question as to their objectionable nature. They are leaky, and offer opportunity for negligent drivers, who by leaving them open divert the current from its proper course or stop it entirely. The most objectionable feature is their liability to destruction by explosion, and the consequent annihilation of the current at the most critical time. In many

collieries an excellent device has been introduced, which consists of the sheet-iron door hinged and suspended from the roof at appropriate places, to be released when the explosion occurs, drop, and close the opening, thus replacing those disturbed by the accident.

Where two currents flow through air-ways which intersect in the mine, the means of deflecting the return air-current is usually an air-bridge and air-crossing. This consists in furnishing in the upper or lower strata a separate air-way and walling up by a permanent masonry or timber dam the area of the return air-way on both sides of the haulage or intake air-way. This opening through the solid rock-called an air-bridge, a crossing, or an overcast, may be lined as a wooden conduit of moderate cross-sectional area, or may be untimbered except for such props and caps as may be necessary to prevent the top rock from caving in. The permanent masonry dams, built on either side of the intake way, are usually from 12 to 24 inches thick, while those which are of timber are made of 3-inch plank, tongued and grooved.

Brattices are used as temporary expedients for subdividing any room or air-way in such manner as to admit of an inlet current on one side to be carried to the breast or face of work, thence returning on the other side of the return air-way. Brattices may be made of planks nailed on props at suitable distances apart, with the interstices between the plank lathed, and the whole tarred or calked with oakum to constitute an air-tight partition, or the brattices may be made of canvas unrolled horizontally and suspended from the roof and frequently adjusted upon posts, according to the ventilating pressure in the air-way or room in which they are placed. To make canvas impermeable to air, it is usually soaked with tar, though its stench has resulted in its substitution by an incombustible material, such as asbestos or a soluble silicate.

Other methods of furnishing a conduit for the return or inlet-way consist in cutting a ditch in the floor of a gallery and boarding it over; in providing a "top sollar," the timbered roof of the gangway having a little extra space for the

passage of the air-current above it instead of below it; in laying wooden air-boxes, metal pipes, or large canvas hose at one side for temporary expedients.

54. EXAMPLE.—A downcast 6×11 , an upcast 6×12 , 300 feet deep, supply air to a mine having a gangway 3000 feet long, 50 feet sectional area (5×10); five splits receive each a portion of the 40,000 cubic feet moving. Required the several amounts delivered to the panels through the resistance of 6×8 , 700 feet long; 5×7 , 1000 feet; 5×8 , 1200 feet; 6×6 , 750 feet; and 4×7 , 800 feet long.

By substitution it will be found that the pressure required to overcome the downcast resistance is 1.23 pounds per square foot. In the mine the pressure will be the same throughout; hence for each split

$$q^2 = \frac{a^3 p}{f l m},$$

which, solved for each division, gives volumes $q = 16,059 \sqrt{p}$; $9030 \sqrt{p}$; $9679 \sqrt{p}$; $10,909 \sqrt{p}$; and $7543 \sqrt{p}$, respectively. The sum of these equals the total quantity Q , from which we get p , equal to 0.87 pound. Assuming that the splits are made as near as possible to the downcast entry and reunion to upcast, which by the way is advised, there need be no further allowance for resistances other than those of sudden turns or contractions. The distribution of the air q for each district is 12,070, 6785, 7275, 8200, and 5670 cubic feet.

The upcast offers a resistance to the exhaust-air, the volume of which is greater than the 40,000 cubic feet, because of accretions from "blowers," moisture, etc. Disregarding these increments, the volume to be exhausted is carried up at a velocity of 555 feet per minute, whence p is 1.00 pound.

The total pressure to be imparted by increasing the downcast barometric or rarefying the upcast is therefore 3.01 pounds, requiring M of 57 feet, or a difference in temperature of about 100° at $B = 25''$. If the splits are made at stated distances along the main gangway, an allowance must be made for each of the several losses of friction in the various lengths thereof, remembering that each branch split reduces the volume passing through the remaining portion of the gallery, and correspondingly the friction therein. It would require 3.7 h. p. to do the work upon the air; a fan of 43 per cent efficiency would necessitate a 9 h. p. engine.

The calculation for the ventilation of a railroad tunnel is similar. Assume a tunnel 4961 feet long; sectional area, 336 square feet. Then $4961 \times 336 = 1,666,900$ cubic feet of air to be changed every ten minutes. Velocity of current, 496 feet.

$$p = \frac{f l m v^2}{a} = 4.8 \text{ pounds per square foot.}$$

A fan 20×6 at forty revolutions easily meets this demand. The fan may be applied, with or without brattice, at either end of the tunnel; but this is a deli-

cate matter. It would be better to place the fan at the mouth of, or a furnace at the bottom of, one of the connecting shafts used during construction, and block the others off.

Required the pressure and power to get 10,000 cubic feet of air through airways aggregating 6300 feet in length, of which 5000 feet is along a heading 5 ft. $\times$ 8 ft. in area, with 1000 feet of airway 6 ft. $\times$ 9 ft. at the intake end and 300 feet of a 2 ft. $\times$ 5 ft. air-course at the upcast end.

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CHAPTER XV.

ILLUMINATION.

55. Use and consumption of candles, etc.; Davy's discovery and invention; description of the safety-lamp; remarks regarding later forms, Stephenson, Mueseler, Hepplewite-Gray, and Marsaut. 56. Requirements of a safe lamp; modes of rendering them secure; candle-power of the different types; electric illumination. References.

55. IN an atmosphere containing gases sufficiently diluted to render it harmless illumination may be had by any form of naked light. In all metal-mines candles are used, and occasionally the torch and kerosene-lamp. In bituminous mines known to be non-gaseous the latter is employed; but in all mines which are at all likely to develop gases the lamp-flame must be protected from direct contact with the white or fire-damp.

Candles which are used in metal-mines are usually of stearic acid, of which Proctor and Gamble's are the most uniform and will best withstand the temperature of the heated atmosphere. They are cheaper illuminators than lamps in rooms and stopes, but not in haulage-ways. The consumption averages three candles per man per shift. The common tin lamp with the hinged lid on top and a hook and spout on either side—from the spout the wicking projects and is warmed—is a more brilliant illuminator, and is also used in coal-mines, giving a moderate light of about four candle-power, with, however, the objection that it smokes.

Kerosene or petroleum is commonly employed as the fluid, but its unsafety requires an admixture of a less volatile oil. White lard, winter-strained oil is also much used, the consumption being one-half gallon per month for each lamp. In some mines rape-seed oil is used, though a mixture of equal parts of seal-oil and petroleum seems best to meet the

requirements of a good illumination with a minimum of smoke. In a mine using 260 duplex-wick lamps the annual expense for oil, repairs, interest, etc., is \$504.00. In selecting oil for illuminating purposes, its behavior is tested not only from the standpoint of its usefulness as an illuminant, but also that of its ability to burn without smoke. When the oil burns and the combustion is perfect, a blue non-luminous heating-flame is produced; but when the conditions are such that the flame is cooled during combustion or receives a deficiency of oxygen, the combustion is imperfect, and the portion of the carbon in the oil is rendered incandescent, thus emitting light. When the oil is very dense, the amount of incandescent carbon released becomes excessive, particularly in the presence of a small amount of oxygen, and soot is the result. The ideal oil, therefore, should furnish a maximum of light and a minimum of soot, with sufficient combustion to produce draught. A simple test and a decisive one may easily be made for the fitness of oil for use in the miner's lamp by burning it, under the ordinary conditions, in a common house lamp with a short chimney. The mixtures, which are often used, of mineral oil with animal and vegetable oil are always objectionable because of the almost unendurable odor, which itself is detrimental to good air. There is little saving in their employment, and they are worse than is either oil unadulterated. The very volatile oils and spirits, like benzine, burn with a clear uniform flame, show an easily perceptible cap in the presence of gas, and are usually very sensitive, being also free from danger in a well-constructed lamp, even in the hands of an unskilled miner.

In mines working under the long-wall system with an ample current naked light may be used, but should be restricted to narrow works, and in mines working by the pillar and stall, to the rooms ventilated by splits, and not for robbing pillars.

A great deal of ingenuity has been expended in the endeavor to invent a safer means of illuminating workings than that offered by the naked flame. In 1815 Davy dis-

covered that a sheet of iron-wire gauze was so good an absorbent of heat that the flame in contact with it could not readily pass through. Further experiments indicated that for mining purposes a mesh of 784 holes to the square inch was the safest, and was therefore adopted as the standard. A cylinder of this mesh, surrounding the light, surmounting an oil-lamp and capped by a perforated top, is the form, which has been little changed since Davy's time (Fig. 108). After the lamp is filled with oil and lighted, it is locked, to bar the miner against access to the flame, the wick of which

is trimmed by a wire passing up through a close-fitting tube from the bottom. The combustion is supported by air penetrating the gauze at all sides.

Sir Humphrey Davy thus describes his invention: "The principle of my lamp is that the flame by being supplied with only a limited quantity of air should produce such a quantity of azotic or carbonic acid gas as to prevent the explosion of fire-damp, and which, from the nature of its



FIG. 108.

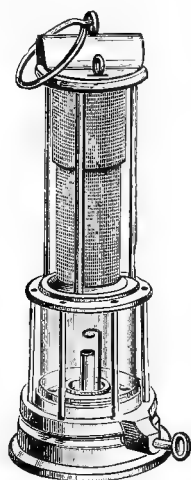


FIG. 109.

operations, should be rendered unable to communicate any explosion to the outer air."

The lamp has done and continues to do great service; but it has two defects. The first is the liability of the gauze to become red-hot, and allow the flame to pass through to the inflammable mixture outside. The second objection is its low illuminating power. The open spaces occupy only one fourth of the area of the gauze, through which the light escapes horizontally; still less light gets out at the top, to illumine the roof. Miners require light thrown in every direction, especially upward; and in a certain investigation,

while giving evidence, confessed that they would rather unmask the flame and risk explosion, than not to watch and see distinctly the roof, the ever-threatening danger of which can scarcely be denied. These defects have been partially remedied in the subsequent patterns by the use of glass, the only impermeable, strong, though brittle, transparent substance.

The Clanny is the first alteration of the Davy, a lower portion of the wire cloth of which, if replaced by a short cylinder of glass, gives somewhat better illumination (Fig. 109). The simple expedient of enclosing it or the Davy in a tin can or shield is also quite an improvement.

Stephenson's, almost as popular in this country as those above, has a long cylinder of glass surrounded by a wire gauze, and bonneted above by perforated copper. The feed is also through the gauze, going underneath and into the cylinder to the flame, thence out at the top, as usual. This plan keeps both cylinder and gauze cool, and its relative security rests essentially on the regularity of the draught, for if the inside air becomes overheated the light goes out; so it must be suspended properly. This is an English favorite.

The Marsaut is an improvement upon this form, and stands a fair amount of tilting safely. With care, its glass cylinder will last three years before breaking. The Marsaut lamp in many mines abroad is regarded as the most suitable one for the working miner, its construction being simple and strong, and as an indicator of gas it is reliable, furnishing also a good light. Of 370 in use, the average consumption of rape-seed oil was 2 gallons per year. This lamp was brought very prominently before the public by the Accidents in Mines Commission. A great difficulty is experienced in relighting it, and from the winding path pursued by the feed air proper circulation does not take place until the lamp gets hot.

The Mueseler, a Belgian lamp, is like Dr. Clanny's, having in addition a conical chimney centrally above the flame. It is highly recommended in Europe, but must be carefully handled. It does not burn well in "dampy" or

slow currents. The bonneted Mueseler, an English improvement, is receiving the highest encomium for use in fiery mines and high velocity.

The Hepplewite-Gray lamp admits air at the top, down four tubes, and through an annular chamber above the oil vessel. The only gauze employed is that covering the outlet and annular inner chamber. A serious difficulty with it is its liability to be extinguished when suddenly lowered. It undoubtedly gives more useful illumination than any other lamp, and as an indicator of gas undoubtedly ranks superior to all others—except, possibly, the Pieler or Wolf varieties. All other forms with the inlet above the glass will miss, say, four inches of gas lying immediately against the roof, except when they are tilted very much, and then there is great danger of their going out. Many lamps are now constructed to take air, if desirable, from the top, like the Gray, and thus also to detect thin layers of gas; but even then they will not do it so rapidly. It is possible to put some modern lamps into gas and take them out again without any indication being given—if the test is conducted hurriedly. This is quite impossible with the Gray, as the flame immediately “spires” up. Owing also to the large amount of useful light given by it and the way this is directed on the roof, in addition to its delicate indications of gas, this lamp is preferred to all others for use by deputies, firemen, timberers, and fire-bosses.

The Dick patent port-hole lamp compels all the air entering the lamp to go immediately to the flame, thus losing no air, and is capable of burning in a stagnant atmosphere. The air entering the lamp above the case passes through the gauze, thence descends to the flame, while the products of combustion arise inside the lamp, to be emitted through circular holes at the top of the bonnet. The bonnet is made of a seamless steel tube, and is light and strong.

The Wolf benzine safety-lamp is an emphatic departure from the varieties above described, in that, first, it burns benzine or naphtha; second, it contains a patent self-igniter

capable of relighting the lamp fifty times without opening; and, third, it contains a locking device which it is impossible to open except by the use of an exceedingly powerful magnet. This lamp, because of the sensitiveness of its illumination, is a delicate detector of gas, and has met with very ready acceptance throughout coal-mining districts, there being possibly 80,000 in use in Germany. Of 18,300 lamps in one mining district in Great Britain, over 10,000 are either Marsaut, Mueseler, or bonneted Clanny.

Notwithstanding the various modifications, there is yet no really safe lamp—one that *cannot* ignite in an explosive mixture outside of it. Generally, the elongated appearance of the flame gives warning of danger to the man carrying it into a fiery atmosphere; and it would be the better part of valor to smother the light or to withdraw from the spot before the heating of the gauze begins.

56. The “safety” lamp must be capable of resisting explosive currents of highest velocity attained underground—that the air-current shall not be able to blow through the gauze into the lamp or to force the flame against the gauze. This permeability is determined by the mesh, and there is a limiting degree of safe coarseness and of speed of current. The Hepplewite-Gray and the bonneted Mueseler have the best resistance to explosive currents of high velocity, and the South Side Committee report the following relative speeds at which the respective lamps and the air-current can safely pass: Davy, 360 feet per minute; Clanny, 600 feet; Stephenson, 780; Mueseler, naked, 1200; Mueseler, bonneted, 2400; Marsaut, in a can, 2440; and the Davy, in a shield, 2400. The North of England Institute of M. E. gives the safe velocities at 720, 540, and the others higher. The British Royal Commissioners of Accidents approved the Gray, Marsaut, and the bonneted varieties as safe at high speeds. The common Davy or Geordie lamps are unreliable.

In order to be safe in the highest velocity of air-currents within a given mine, the flame must be enclosed not only in a wire gauze, but also in a more or less impermeable hood or

bonnet, while the inlet area for the feed-air must be reduced to the smallest allowable dimensions. Many lamps now exist which appear to resist, in a highly explosive atmosphere, current velocities up to 3000 feet per minute for a period of several minutes; and the four lamps which were brought to the attention of the Mines Accident Commission, which received special attention for their security, illuminating power, and simplicity of construction, were the H.-Gray, Marsaut, bonneted Mueseler, and Thomas's modification of the bonneted Clanny.

The bonnet screens the gauze cylinder from the effects of draughts that blow the flame through the meshes and set up a fiery heat by the excessive air and gas that enter above the flame of the wick. It limits the supply of air to that required for the oil flame only. Such bonneted lamps, whose flames are protected from the direct effects of the strong ventilating current, may be used with safety for illumination in mines producing fire-damp. Even in dry, dusty mines also developing fire-damp some of these lamps are safe, though not all; for many well-authenticated cases of failure are recorded where the dust has proven fine enough to pass through the gauze meshes, to be reduced to the state of incandescence in the inner chamber.

Of the forty-one explosions which occurred in a certain district during 1896, in four cases the immediate cause of ignition was referred to a naked light or to a deterioration of the safety-lamp; in twenty-five, to the passing of the safety-lamp flame, in consequence of the gauze heating through a careless movement, too high a speed, or "falls in." The remaining twelve were from shot, fire, or other undetermined causes.

The importance of locking the lamp so that its flame cannot be exposed to the gas is manifest, as there are many temptations to the miner to open it in order to better illumine the roof or to light a pipe. Either practice is reprehensible. All manner of permutation-locks and magnetized plates are offered on the market, besides the lead-plug seal with which

the lamp is riveted after each filling. The latter is giving satisfaction in S. Wales. The magnetic locking device of the Wolf lamp has proven effective to resist all efforts of the miner to open it. Other lamps are so constructed as to extinguish the flame when the oil vessel is separated from the gauze cylinder.

The illumination from any of these lamps is very feeble—best horizontally, but less in any other direction. Of all the lamps the Gray sends the best light upward. The candle-power, horizontally, of the Roberts is highest—about 18, and of the Clanny the lowest—nearly 6. On this account a lamp must be able to be held tilted without extinguishment, and be unaffected by violent oscillations. The conditions dictated by safety circumscribe the lines of attempted improvement in the degree of illumination. The brass lamp is found to be 70 per cent as bright as the iron lamp of the same pattern preferred by the Germans. Photometrically speaking, seal-oil is better than rape-seed, and a broad, flat wick than a round one. The insufficiency of the light of a safety-lamp, combined with the difficult and trying conditions of the bonneted forms, is proving injurious to the eyesight of miners, which serious evil is growing. Photophobia is rare where candles are used, or where the lamp is hung behind the miner.

At Zwickau, Saxony, a novel and bold plan is in use, owing to the difficulties with all safety-lamps; an innumerable quantity of naked lights are burned constantly, which ignite the gas as fast as it reaches the candles. No explosions have been recorded.

Whatever the means of illumination, the lamp must be self contained, be strong, portable and not heavy, require little attention from the miner during twelve hours of sustained light, and be capable of placing in any position, besides giving perfect insulation from the fiery gas. In an incandescent lamp, wire-bound, and with flexible connection, electricity fulfils many of these requirements, besides requiring no oxygen, and it seems reasonable to expect it to supersede the present form of lamp. Its success in metal mines makes the proposition for collieries not so absurd as would at first sight appear. Large

chambers would thus be safely and so thoroughly lighted as to render every part of the roof visible, affording greater security to the hewer. A greater number of lights would be required than of oil, as the former cannot be continually carried about beyond the limit of the flexible connection. Again, along the entire galleries numerous lights would have to be placed, except in the haulage-ways, where lamps in the hats may be permitted. Though the electric system is not sufficiently perfected, many mines employing this force for other purposes find it better and not much dearer than oil. The cost of a plant for 100 lamps, exclusive of the generating machinery, is \$500; and for coal, renewals, interest, etc., the annual expenses are \$518. One h. p. will run ten 16 c. p. lamps at 75 to 150 feet apart. The life of a lamp (60 cents) is fully 100 shifts. A serious detriment is the fracture of, or the injury to, the wires. A portable, self-contained secondary battery lamp may obviate this, but it is both heavy and wasteful of power.

Lamps are not safe unless kept in thorough repair, and infractions of rules regarding their use severely punished. The gauze should be steeped in a hot alkaline solution, to free it of soot, etc. Lamps burning benzine are not clogged with carbonaceous deposit as are those burning oil. There is

To avoid waste, manufacturers furnish automatic fillers, holding just enough for a lamp. Lamps should be occasionally tested for leakage and other sources of danger.

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CHAPTER XVI.

HYGIENIC CONDITIONS.

57. Laws upon ingress and egress; accidents in mines; ladders, their arrangement and cost; loss of time and energy; use of cages for men; conclusions of the Cornwall Society. 58. Movable ladders or man-engines, single or double; utilization of the pump-rods for the purpose; comparison of the safety of the man-engines with other means; cost of the machinery and plant. 59. Accident laws for the protection of life and limb; are equally effective for the security of the mine; statistics; accident-rate decreasing; tables; lessons drawn from their inspection; causes and prevention of accidents; fall of roof; lack of timbers; explosions; premature blasts; necessity for a rigorous enforcement of the rules and laws. 60. General remarks concerning fires in mines, their causes, prevention, and treatment; entering old mines; aërophores. References.

57. FOR purposes of ingress and egress, mines are provided with ladders or man-engines, where the cage or bucket is not used. The statutes of many States present varied ideas, theories, and requirements for the accommodation of the men. Some require the maintenance of substantial ladders in a separate compartment, as the sole means to be used by the men for entry and exit. In other States operators are relieved of the necessity of keeping up a ladder-way, if safety carriages are employed. The laws of many States forbid the use of buckets by the miners, while the general tendency in all regions is to insist upon two well-equipped escapement ways.

If the angle of entry is below 30° , no special provision is necessary. The mud-sills of the timbering break the descent into sufficiently convenient steps. Steeper than this, and up to about 60° , some variety of treads is necessary. When the pitch exceeds this, the compartment must be provided with ladders, isolated from the hoistway. They should be inclined,

uniform in direction, at an angle of not less than 10° from the vertical, to diminish the fatigue of climbing, and enable the men to carry tools with them. At equal distances down the ladder-way (20 to 40 feet down a vertical shaft, and at greater distances on an incline), platforms are built of 2×6 beams and 2-inch planks, closing it, except for a man-hole, at the foot-wall end. The ladders extend up through the man-hole, and are fastened by staples or toe-nailed to the shaft-timbers, and rest on the far side of the plats. They are made of 2×6 standards, 18 inches apart, with iron or wooden rounds or rectangular slats 12 inches apart. The last-named are cheaper, last longer, and give better toe hold than wooden rounds, which, in turn, are easier to use than the more durable iron. Wooden ladders cost from 6 to 10 cents per running foot; iron, 20 cents.

Though used in Europe for 1200 to 1500 feet depth, and in this country in deep mines, they are certainly not advisable. According to the Cornwall Society, the use of ladders deranges the respiration, and shortens life by ten years. The miners reach the workings more or less exhausted, and the operators have lost the benefit of a proportionate amount of energy. Unquestionably, an element of success worthy of attention by mine managers—a pecuniary as much as a humanitarian question—is the proper treatment of and the conveniences for the men, who unconsciously reciprocate in an equivalent of work. Besides, time is lost. It takes 15 minutes to go down 300 feet, and the ascent is twice as slow. A shift of forty men, following one another at intervals of 8 feet, entails a loss to the company of 31 minutes each shift. With buckets and cages the loss is not so great; eight men at a time, lowered 1200 feet, consume 40 minutes for every shift of 100 men. An additional loss occurs at tally-time from the reduction of the hoisting capacity, which, with the impatience of the men, leads to the crowding of the cage; but in most States the limiting number of men permitted on the cage is named. A serious form of accident, peculiar to deep mines like the Comstock, is the fainting and falling, which occurs when the heated miner,

while being hoisted, comes into contact with the air near the surface. There is no safeguard against it, and owing to its frequency men never go up alone.

58. Movable ladders or man-engines, invented by D'Orrell, of Clausthal, were instantly adopted as acceptable substitutes to the methods previously used, and now are very popular in deep mines. Mr. Lorn, who introduced the engine in Cornwall, was handsomely rewarded by the Royal Polytechnic Society, which declared it a "great boon to miners." Its introduction involved the addition of some machinery, but it was easy to operate.

Two rods, of decreasing cross-section from top down, receive at the surface an oscillatory motion from balanced bobs, operated by an engine having a fly-wheel and other regulators. The dimensions of each rod at any point must be such that it will have the requisite tensile strength to support the weight of the part below it, loaded with men. They play between roller-guides 50 feet apart, and are provided with wings and catches, after the manner of the Cornish pump-rods, which may, in fact, be utilized as "Fahr-kunst" rods without much extra power.

Each rod has a small platform, Fig. 109, about 12" $\times$ 12" or 18" at every 12 feet—double the length of the stroke. A handle four feet above the platform gives support to the miner, who is carried up 6 feet on one rod, which brings him opposite a platform on the companion rod; upon this he steps, to be lifted 6 feet more, to meet a plat on the first rod, which has been coming down to receive him. A miner stepping from one to the other is carried up or down at a rate of from 48 to 96 feet per minute (each rod makes 4 to 8 double strokes, delivering one man each

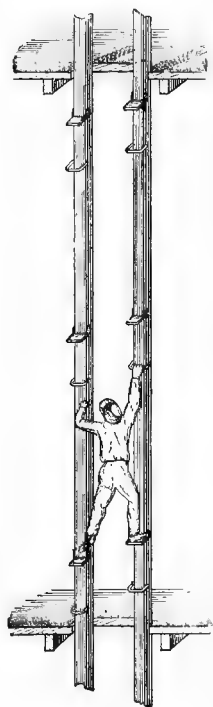


FIG. 109.

time, those at the Calumet and Hecla make five strokes). As there is no limit to the depth at which these may be carried, and as they are capable of working alike in slopes as in shafts, it is not surprising that they "take" so well. They replace hoisters, and require little additional power or space. Tools and supplies cannot be carried by the miner, but may be delivered by the cage or bucket.

A single rod is also used, its companion being replaced by stationary platforms attached, 6 feet apart, to the shaft timbers. Upon these the ascending men wait during the down stroke of the rod. The single-acting man-engine requires chains and counterpoises at intervals to balance it, and to prevent the shock incurred at the end of the stroke.

From the fact that a misstep would be fatal, it would seem as though man-engines were extra-hazardous, yet the accident record does not confirm this fear. Some confusion is caused by a man missing his plat and riding on, to the annoyance of those following him; but this is of rare occurrence unless his light goes out, for there is a halt of several seconds at each change of motion. Out of an average of 100,000 men employed for ten years (in Prussia), only 57 were injured on the man-engines; in Cornwall, 17. This is more than compensated for by the increased length of life of the miners using them.

The cost of machinery, etc., for a 1200-foot man-engine is \$18,000, upon which interest and depreciation may be figured at \$2500,—amounting to 10 cents per man daily, on a gang of 100 men. The running expenses at the Dolcoath mine are 4 cents per man, 2400 feet.

59. We now arrive at the consideration of a theme which, sad as it is, should suggest the lines of improvement. Deplore as we may the immolation attendant upon mining, there seems no way, by legislation, threats, or punishment, of impressing the necessity of vigilance upon the miners, who by long inurement to peril that is imminent have become oblivious of the unavoidable sources of danger.

The statutes make stringent requirements of the operators and of the employees, enforce frequent thorough inspection.

by competent men, impose fines and penalties for negligence or non-compliance, and our appliances are useful, durable, and modern; yet the benefits that should accrue are not realized—the death-rate continues deplorably high. The percentage of accidents in steep vein-mines is less than that of iron-mines, and only half that in coal-mines, where 3 out of every 1000 employees are injured annually. Bituminous collieries are more dangerous than anthracite or lignite mines. The rate has been decreasing somewhat, as might be expected, though the increased depth of working tends to make mining more hazardous; and, assuming equal conscientious announcements by the authorities reporting the casualties, it will be found that the safety of life in our mines bears satisfactory comparison with that in European mines. Generally, the accident statistics are compared with the output tonnage, and it may be said that for every 200,000 tons of coal mined one life is sacrificed and two men injured. That this proportion is diminishing is patent to any one inspecting the reports of inspectors. Though it is difficult to get a trustworthy comparison of the number and class of accidents, the following table is given, showing in percentage the fatalities and casualties. The miscellaneous accidents vary from 2 to 27 per cent of the total number.

	Falls of Roof and Coal.		In Haulage Ways.		Fire- damp.		Powder.		Tons per Accident.	Tons per Life Lost.	Employees per Death.
	<i>f</i> *	<i>s</i> *	<i>f</i>	<i>s</i>	<i>f</i>	<i>s</i>	<i>f</i>	<i>s</i>			
Pennsylvania, 1889:											
Anthracite	45	32	25	13	6	16	7	11	34,817	105,764	342
Bituminous	65	60	22	25	5	2	3	1	102,414	397,612	585
Illinois, 5 years	63	60	11	11	4	3	5	14	66,200	215,549	716
Ohio, 1874	61	58	14	18	11	9	4	3	4,844	108,919	412
" 1889	76	61	15	21	..	2	6	5	128,322	330,529	619
Iowa, 1889	64	63	17	15	8	6	5	7	52,400	98,620	321
Missouri, 1889	50	..	..	6	..	10	..	13	111,173	222,347	514
Nova Scotia, 5 years ..	62	64	18	16	9	7	3	4	167,083	238,697	618
Comstock	..	12	..	21	..	..	..	8	52,140		...
Missouri, zinc	..	45	..	19	..	..	..	5	8,376		...
Colorado, ore	..	60	..	21	..	..	..	9	19,460		...
Illinois, 1889	70	67	13	20	7	2	8	5	42,988	238,450	539
Italy, 1889	54	61	24	19	8	11	9	2	\$42,400	\$152,993	752

* Columns headed *f* are fatal accidents; those headed *s*, serious.

PERCENTAGE OF FATALITIES.

	Falls.	Ladders.	Fire-damp.	Powder	In Shafts	Tons per Life Lost.	Employees per Death.
England, 1851.....	34.3	1.7	30.0		21.2	63,562	222
" 1888.....	53.0	2.3	5.4	4.0	8.5	194,430	600
Prussia, 1852.....	50.0		13.5		20.4	83,051	600
" 1889.....	40.0	2.0	12.8	5.0	10.0	109,528	365
France, 1853.....	40.0		20.1	3.6	21.4	37,346	260
" 1889.....	30.0	1.5	28.6	1.0	14.3	117,105	538
Belgium, 1861.....	36.7	2.1	10.3	3.1	13.4	51,840	422
" 1888.....	35.3	1.5	16.0	2.8	8.2	106,110	571

But the tales which these figures tell must be noted. First, that notwithstanding the frequent holocausts, with the reports of which we are shocked, the loss of life by explosions and fire is not by any means as great as by the more numerous unpublished accidents to individuals resulting from the caving of roof by reason of insufficient timbering. Fully one third of the deaths are from this cause,—and the percentage was the same in the '50's as now—and neither the operators nor the bosses are responsible always for them, as subsequent investigation reveals. The crushing of men by the fall of coal upon them is an equally common accident. Many casualties are caused by the indifferent miner, anxious to make a big turn-in, neglecting to support the roof of coal with the timbers right at hand; in fact, I have seen instances where a crush had caught victims who were compelled to crawl over a supply of props in order to reach their work. It is an incontestable fact that the miner will take too many risks, and an accident ensues solely from his own carelessness. It would be unjust to attribute all accidents to wilful neglect, for mining is precarious; but surely many calamities might be avoided if the miner would exercise precaution. It is not sufficient that he is the victim of his own wilfulness,—for the evasion of the law carries its own penalty,—but he endangers the lives of his co-laborers, and the property of the employers who have invested heavily in measures for his protection. The sudden dislodgment of the roof or sides of a breast or stope, or the unnoticed yielding of the pillars, is due

The various grades of underground accidents which occur in collieries and in metal-mines bear nearly the same ratio to one another in the two classes of workings, though the aggregate number of accidents and fatalities may not be, in the two cases, the same. A comparison of the lists of fatalities of the earlier periods of mining in the current century with the lists which are published annually at the present time demonstrates the great improvement which has been effected in underground conditions; many of the evils surrounding a miner's life in the early days have been removed, while the consequences of the other sources which remain have been lessened to an extent that makes the occupation of a collier more than tolerable; indeed it is no longer the most hazardous.

That coal-mines are more hazardous than metal-mines is commonly but erroneously believed. For, while the number of injuries or fatalities is small in the latter, yet it must be borne in mind that the number of employees is also *smaller*, thus making the proportionate number of injuries from accidents in coal-mines lower than in metalliferous mines. The latter class of mines are under less rigorous inspection than are those of the former, in which the danger from explosions of fire-damp has been so far reduced as to almost eliminate this cause.

The sudden dislodgment of the roof or sides of a breast or stope, or the unnoticed yielding of the pillars, is due to so many causes, that it is impossible to prescribe rules for its **prevention**. Horses, sigillaria, balls of ironstone, rock creviced naturally or by excessive blasting, are threatening conditions that demand a liberal supply of precautionary timbers or filling placed before the movement begins; otherwise, once begun, no amount of subsequent support will save it: the ensuing damage is out of all comparison with the insignificant item of props judiciously used. Moreover, without a better system of illumination of the underground workings the miner cannot discern the condition of the overhanging rock, and props, to be opportune, must be placed at once. The substitutes of iron, steel, and masonry for wood must conduce to a greater safety, as also the increased facilities for the more expeditious removal of mineral.

Falls of roof are responsible for the great majority of lives lost underground, whether in coal or metal mines; and, while this class of accidents is most frequent in thick seams and steep seams, nevertheless the occurrence of weak spots and "bell-moulds," suddenly liberating masses of stone without warning, are equally frequent. The fall of coal from the side, during the long-wall or pillar and stall working, also raises the mortality in coal-mines, the remedy for this, as for the others just mentioned, being a plentiful supply of timbering set with their joints at right angles to the lines of cleavage, and limitations in the size of the excavation. If the miner would be more watchful and promptly set the timbers this grade of accident would diminish. It is always the so-called safe roof of overhanging rock which causes the trouble. Men, who would never think of opening a safety-lamp will continue to labor under a roof which, they think, will stand till night.

The sole remedy, in the opinion of the author, lies in the issuance of an order for the unconditional dismissal of any employee who fails to prop loose rock, and a rigid, instantaneous enforcement when the discovery is made.

A brief study of the table of deaths in the mines of the United Kingdom during the history of coal-mining verifies the conclusion that is quoted of our Pennsylvania Mine Inspector, that accidents from this cause and "from cars are regular and uniform items in the death and injury lists, whereas those resulting from blasting or gas-explosions are sporadic and irregular."

Arranged by decades, a similar table of accidents for the anthracite fields of Pennsylvania would give equal results, though to attempt the comparison of accidents with other countries is difficult, because of a difference in statutory requirements, the lack of uniformity in the definition of "fatality," and the imperfect classification of accidents. Mine inspectors give preference to the place of occurrence rather than to the cause of accident in their classifications and reports.

AVERAGE NUMBER OF ACCIDENTS IN MINES OF THE UNITED KINGDOM IN SEMI-DECADES.

	Number of Em- ployees.		Average Annual Fatalities.				Total.		Death - rate per 1000 Persons, all Causes.	
	Below.	Above.	Falls.	Shafts.	Gas-explo- sion.	Miscel.	Below.	Above.	Below.	Above.
1851-55..	182,427	47,047	368	235	231	101	937	47	5.149	1.012
1856-60..	208,763	53,832	385	187	257	135	964	53	4.628	0.994
1861-65..	237,779	61,314	406	159	146	186	898	68	3.791	1.105
1866-70..	269,813	69,574	426	142	306	196	1071	87	3.995	1.256
1872-75..	399,397	111,584	470	174	195	225	1066	99	2.736	0.899
1876-80..	424,586	117,876	480	134	341	190	1147	100	2.709	0.847
1881-85..	443,502	116,688	491	116	181	236	1025	99	2.312	0.848
1886-90..	477,633	126,654	482	93	152	247	975	117	2.042	0.913
1891-95..	571,463	150,804	457	111	141	270	979	123	1.705	0.822

Accidents in and about the traffic-ways are being reduced by the use of safety appliances, previously referred to; by gates and doors at the mouth of shaft and level; by a small drift cut in the hanging-wall, for miners to pass around instead of across the shaft; by whitewashed safety niches at every 100 feet in a gangway; by care in signalling; and by having a space 2 feet wide between the "loaded" cars and the side of the heading.

The effects of sudden changes of temperature experienced by those coming from a hot portion of the workings to the surface may be remedied by prudence on the part of the miner, and by railings around the cages. The new Pennsylvania law requires hand-rails on cages. Dr. G. C. Swallow, Mine Inspector of Montana, suggests an excellent idea to prevent the mutilation of men riding on the cage. A coiled wire screen, which may be drawn down at the sides of the cage, is fastened below, and prevents the contact of men with the timbers. Except in fainting, men caught between cage and timbers have only themselves to blame for accidents on cages.

Miscellaneous accidents in shafts arise from materials falling therein, or from over-winding, or the breaking of hoist rope or chain. The only remedy for this is the maintenance

of fences at the surface, and a careful daily inspection of ropes, with a simple and unmistakable system of signals.

Accidents caused by premature blasts are more frequently the result of carelessness, though many unaccountable explosions have occurred. Electric firing of cartridges, the prohibition of loose powder, and the avoidance of firing in collieries while much fine dust is afloat materially diminish the casualties.

After all, neither legislation nor appliances will avail if the men do not have an ever-present realization of impending danger, and a corresponding caution. Doubtless many of the charges of carelessness are unjust, for only at the critical moment may have come an instant of absentmindedness, when the fatal act was committed. The only hope is for a change in human nature, for until men willingly obey the laws, and on occasion deny themselves of some slight fancy, accidents cannot entirely be prevented. All precautionary measures should be announced, rigorously enforced, and the offender, in no matter how slight a particular or what the plea, discharged. It is not the visitors who are the victims: it is the old hands, in the pockets of whom pipes and matches are presumptive evidence. These, with fuse, tobacco, etc., should be contraband goods, and subject the miner introducing them to fine.

The great mortality in metal-mines, as compared with coal-mines, is unquestionably from carelessness when we find on the list, among the common causes of accidents, "thawing frozen dynamite," "igniting too many blasts at one time," "drilling into unexploded cartridges," "using iron tamping-bars," and "tampering with the metallic caps."

Before a recent Austrian Government Mining Commission (1897) evidence was given that, of the accidents that have occurred there 60 per cent were caused by the victims or their mates as the result of carelessness or incompetence; 30 per cent were caused by circumstances over which the men only had control; and the remaining 10 per cent were of the non-preventive class. This general deduction would likely be verified by those cognizant of the facts in other districts.

But as to the character of accidents which are to be termed preventable we are not assured. Perhaps only those not anticipated are non-preventive.

The causes enumerated thus far involve usually only the direct instrument of the accident, or the mines in the immediate vicinity of the accident. But there are causes more or less preventable which, when casualty does occur, involve a multitude of lives; chief among them may be enumerated as follows:

1. In the opening out of a seam beyond a fault. This frequently gives rise to a slip in the ground, if soft, and always the evolution of a large bulk of gases.
2. The working of a piece of coal in advance of the face, which likewise delivers a heavy outpour of gas in the shape of blowers that are liable to be converted into dense accumulations at any time.
3. Walling in gases. It was a very general custom to wall in the gob in many collieries, but the result has been that with a sudden and extreme change from high to low barometer a great volume of gas was released into the mine. Moreover, as air could not be entirely isolated from the chamber, it was impossible to prevent the formation of a mixture more or less explosive. All such works were performed at great risk; and the present system, therefore, is to leave all those places open, keeping them constantly clear of all accumulation.
4. Shot-firing in main intakes. The danger here arises from the quantity of dry coal-dust usually produced not only by the blasting, but also pulverized and distributed by the cars. A thorough wetting of the surrounding surface will afford the necessary protection before firing each shot, unless flameless explosives with the use of electricity be used.
5. The firing of goaves. In these spaces the coal-dust, slack, gas, and pyrites are stored; and with the inevitable spontaneous combustion that follows, a pressure ensues with greater or less "breathing," by which much gas is exuded through the fissures in the bottom, at the roof, or from cracks in the side walls. When this source of gas is not recognized, the danger ensues from the impoverishment of the air, even if the tem-

perature of the mass within the goaves does not reach that of the point of ignition of the material contained therein. Undoubtedly the goaves should either be imperviously walled up or entirely removed. 6. Approaching water. The nearest distance that it is safe to work towards water varies according to the thickness and the nature of the seam, the width driven, the width of the place containing water, the difference in level of the two places, and the pressure and volume of the confined water. Defects in the shaft-tubbing, the breaking of the strata under a reservoir and above the mine, and the cutting into the old workings filled with water, cause many accidents under the general name of inundations. The irruptions from abandoned works are the result of a breach of rules regarding reservoirs and bore-holes, or are due to old inaccurate plans. A remedy for this is a due precaution in the establishment of barrier-pillars. Any seepage or appearance of "bleeding" at the face should be traced.

Spontaneous combustion can be obviated only by an active air-current, or by change in the method of mining to one involving complete removal of the coal and a substitution of clean waste. Gob-fires and accidents therefrom are eliminated by removing the aggravating source, oxidation.

A sudden disengagement of gas into abandoned workings not copiously ventilated, or into those in which the air is stagnant, may lead to fire by spontaneous combustion, or to explosion if the necessary conditions are present. The means calculated to prevent accident from such irruptions of gas comprise two classes of measures: those having for their object the prevention of the outburst, and those intended to protect the men from the disastrous consequences of the outburst. A gas-tight dam or bulkhead is often built as a stopping to goaves. Bore-holes in advance of the work, required by law in many States, constitute one means of safety for the mine by giving timely warning of danger. Nevertheless the difficulty of boring them in disturbed, and consequently infested, portions of the seam is not easily overcome. The practice is sharply criticised by many.

Bore-holes from the surface to the gaseous portion of the seam are of common occurrence in the several coal-fields of Pennsylvania. This is the most effective of the remedial measures. When several seams are simultaneously worked, the least gaseous one is driven in advance of the others. Another remedy is based on the discontinuance of heavy blasts of explosive after the men have vacated the mine. These dislocate the measures, cause fractures therein, and tend to release the fire-damp. If the danger is not great, the method is recommended as economical and admits of a rapid advance of working-places.

The measures which may preserve men from the consequences of sudden outbursts of gas include the use of safety-lamps; the use of vertical air-partitions dividing the two currents coming directly from the downcast or going immediately to the upcast; and the installation of a plant for the rapid evacuation of fire-damp by wide and multiple galleries and large fans. A service of compressed-air pipes led into the numerous districts and supplied with cocks and taps is a perfectly rational method to employ for the increased safety of the men.

The explosion of gas, with or without dust, which follows its ignition is second to "falls" in the number of men involved. The nature and extent of some of the most prevalent conditions preceding it have long been understood. Nothing but eternal vigilance and anticipatory action can decrease the magnitude of the fatalities from this source, and the disaster following it can only be guarded against by an active air-current, and the exercise of precaution by stout primary and secondary doors properly hung. A liberal interpretation of, and a willing compliance with, the section requiring two outlets will bring its own reward. But co-operation must be had from the men, as neither laws nor improved appliances can counteract the effects of their recklessness.

The average percentage of accidents from this cause for the past twenty years in Pennsylvania is 0.35 man per thousand

employees, with a marked decrease during the second decade. Some seams give off very little fire-damp, and consequently a very moderate supply of fresh air is required to dilute it. But even such seams as are said to be "non-fiery" should be worked with safety-lamps, unless, they give off sufficient water to moisten the atmosphere and prevent floating dust. In the first annual general report of C. Le Neve Foster, for Great Britain, his classification as to causes of explosions of fire-damp or coal-dust during 1894 shows that out of a total of 189 persons 152 were injured by explosions due to naked light or imperfect lamps, 16 by shot-fire, and 14 by the accidental or spontaneous ignition of mineral or other material. The causes which have led to the gradual decrease in injuries and fatalities from explosions are, no doubt, better ventilation of the mines, both by the increasing volume of air obtained by mechanical means, and a better application of the air to the various faces; but there remains much to be done to diminish the frequency of explosions, by further use of bonneted lamps carefully inspected and of safety explosives. The disastrous explosions of recent years which have been caused by shot-firing are due to the secondary ignition of the gas by the flame blown out from the blasting-agent, the explosion being more or less aggravated by the presence of floating dust. The reduction in the amount of blasting during the past years, and the almost entire cessation in the use of black powder in some mines which are liable to produce gas, has led to a steady diminution of fatalities in those districts. Mr. Foster has very graphically shown in the report quoted above, by a series of diagrams, a marked decrease in the death-rate per thousand employees, not only from explosions, but also from all other causes which involve underground employees in the mines of the United Kingdom from 1851 to 1894.

In an article by Mr. Garforth on the recovery of coal-mines after explosions, he remarks that it seems somewhat strange that the mining world does not to-day possess a code of rules which would be of practical use in the time of great

excitement and confusion, such as usually follow a colliery explosion. Among the suggestions for precaution to be taken by the manager before the accident are the following:

1. Consider the quickest and safest mode of descending into the mine when the usual winding arrangements are useless.
2. Plan for the installation of a special winding engine.
3. Connect water-pipes from the surface to the mine, with branches for use in case of fire.
4. Arrange for the fitting of an extra engine and fan for the emergency.
5. Keep the tracings of all working plans to within three months of work, showing all roads then open, the position of overcast, doors, and brattices.
6. Accustom the men periodically to travel certain roads which they are not in the habit of taking, having finger-boards showing the direction of the upcast pit.
7. Keep on hand all safety apparatus, including Fleuss machine, a quantity of light air-pipes, and "first-aid" appliances.
8. Appoint, during ordinary working of the collieries, some of the leading officials to act as emergency officers at the time of accident, drilling them in their duties.

Suggestive rules for guidance after the explosion are also presented by Mr. Garforth:

1. Send for the emergency officers, assign to each his duty, and appoint some one as deputy in the event of serious accident to the manager.
2. Examine all old connections with the shaft and arrange to repair broken stoppings. Prepare stretchers and stimulants and arrange for a hospital.
3. Provide exploration parties of five with leaders, supplied with safety-lamps, mine-plans, restoratives, cylinders of oxygen, and a stout cord. On no account must any one enter alone, even on the shortest journey.
4. In advancing, the party should move in single file, the leader of each search-party alone testing for gas. Do not let a safety-lamp be the final guide as to the absence of after-damp.
5. Loss of life to explorers may, perhaps, be avoided by remembering the dangers: (a) after-damp; (b) falls of roofs and sides; (c) underground fires and consequent risk of a second explosion.
6. If the force of the explosion has blown out the separation-

doors and overcast, they should not be restored because of the possibility of undiscovered fire. 7. Main intake air-ways, blocked by falls, must not be traversed except by carrying with them an unrolled brattice-cloth, which will admit of a double current of air through the air-way. The brattice should be non-inflammable. 8. To discover the existence of fire, restore the ventilation and examine the return air-ways every hour for (a) fire-stink, and (b) a rise in temperature. If the former be noticed, that section from which it comes should be explored first, and its fire extinguished if possible, or it should be closed off by stoppings, or, in extreme cases, the pit entirely closed. 9. Parties should be careful not to go too far at once, even when taking air with them, as the force of the explosion will have forced the after-damp into the interstices of the goaves, whence it will gradually exude.

60. The causes of fire are quite numerous, and cannot be always avoided. If the surface plant is not placed so precariously as to imperil the shaft, the causes, primary and secondary, are careless blasting, insecure safety-lamps, inadequate ventilation, and floating dust. The first three are most frequently responsible for much of the danger. Explosives whose temperature of detonation is less than 1000° F. are incapable of igniting fire-damp, unless the holes are badly stemmed. Unfortunately very few available compounds realize this condition, unless it may be ammonite. So the only security lies in an almost instantaneous mixture of the deflagrated gases with an ample supply of air, which after all is the only preventive of fires. One of the consequences of the replacement of the hewers by machine is that blasting, as the other operations of mining, falls into the hands of a specialist. This diminishes the accident rate from "shooting." A fuse burning without flame is essential; a means of lighting it, without the fear of sparks which are first thrown off, coming into contact with the air, is obtained in the Heath & Frost lamp. The powder flame cannot be entirely suppressed, even by tamping with water. So the substitution of electric firing, at tally times, for the practice of single shots, is about the only

other means of lessening the risks. It is in the driving of the main levels, winzes, and upraises (preparatory works) that the dangers of the fire-damp are the greatest, because the escape of gases is strongest from freshly cut coal; capillary fire-damp is not difficult to manage, but that under pressure at great depths is serious. Often, near clay veins, the danger of igniting blowers by shots may be avoided by drill-holes, kept in advance of the drift.

It is now conclusively established that soot is a provocative of fire. The effect of the presence of coal-dust has been a subject of trial and many experiments; the most recent—those of Wm. Hall at the British Home Office agreeing with those of the Prussian Fire-damp Commission—show that, without a fierce flame from a blown-out shot, coal-dust in absence of fire-damp might not explode: that it could not alone originate an explosion, though, if initiated by fire-damp, soot may aggravate its effect. This will depend upon the degree of its fineness, the readiness of its diffusion, and its chemical composition.

Coal dust presents conditions but little less perilous than “damp” in mines, but precautionary measures for it are simpler than for gas. It intensifies and extends an explosion originated by gas. Without any floating dust, the flame from a blown-out shot does not travel more than 25 feet, but soot may convey the flame even 200 feet. A means of laying the dust, developed in some diggings, is by a spray continually delivered to the air-current from 1" or 2" pipes under 50 pounds pressure, and a stand-pipe 3 feet high at every 50 or 70 feet. The spray is delivered through lead plugs, slit as desired. No other jet or fibrous material gives an equally fine spray.

Our knowledge of the influence of coal-dust and its effect in the mine on explosions is very meagre, no important commission having been appointed to supplement the investigations of the two great bodies which have already conducted such valuable researches in this general line; but the following brief account of the rise and progress of the coal-dust theory is summarized from the Final Report of the Royal Commis-

sion, which, after discussing these various points *seriatim*, and giving details concerning several of the great explosions that have taken place of late years, reaches the following conclusions:

I. The circumstances of many explosions, and especially of explosions on a very large scale, and covering a great length of the workings, cannot fully be explained by reference to fire-damp or gas alone.

II. The presence of coal-dust, and especially of fine dust, may be the sole cause of an explosion.

III. If the coal-dust is in sufficient quantities it will certainly extend the effect and increase the intensity of an explosion caused by any other means.

IV. Fire-damp in small quantities, so small as not to be dangerous *per se*, may be highly dangerous in the presence of coal-dust.

1. The danger of explosion in a mine in which gas exists, even in very small quantities, is greatly increased by the presence of coal-dust.

2. A gas-explosion in a fiery mine may be intensified and carried on indefinitely by coal-dust raised by the explosion itself.

3. Coal-dust alone, without the presence of any gas at all, may cause a dangerous explosion if ignited by a blown-out shot or other violent inflammation. To produce such a result, however, the conditions must be exceptional, and are only likely to be produced on rare occasions.

4. Different dusts are inflammable, and consequently dangerous, in varying degrees, but it cannot be said with absolute certainty that any dust is entirely free from risk.

5. There appears to be no probability that a dangerous explosion of coal-dust alone could ever be produced in a mine by a naked light or ordinary flame.

Some mine-fires are started in the stables, pump-room, or at oiling-stations. The prohibition of naked lights, a care in handling the oil and waste, and a liberal use and renewal of clean sand and gravel absorbent are recommended.

Sometimes, when blowers of marsh-gas ignite a mine, wet cloths will beat out the fire. But when it has attained such headway as not to be overcome by ordinary means, it may be effectually confined by cutting off the air-supply and building masonry dams, completely stopped up, if the superincumbent strata are not porous, or the mine is not so shallow that air is admitted or the gas escapes. This failing, the burning portion is hermetically sealed, and then drowned with water, or better, CO_2 . For extinguishing the fire at the Calumet and Hecla (supposed to have been communicated to the shaft timbers by the friction due to the binding of the rollers on which the hoist-rope rested) all sorts of plans were resorted to: among others, the surface was kept frozen to stop leaks; finally, the shafts were sealed and CO_2 injected. For the manufacture of 350 cu. ft. of CO_2 there were used 1200 gallons of sulphuric acid and 4500 lbs. of limestone. Water is the simplest quencher, but it has happened that the water could not reach certain portions of the mines above the foot of the shaft, because of the compression of the air which could not escape. Until it was consumed, the fire continued to rage above the water-level, perhaps for a long time. On pumping out the water the conflagration might break out again. A pipe leading from the face of the burning portion, up the shaft, would release the air and permit quenching.

At the Anaconda mine, Montana, steam was injected into the burning stope, but it failed to quench the fire.

For penetrating a very impure atmosphere aerophores of different makes are to be had. They consist of a portable bag or cylinder carrying enough compressed air or oxygen for the respiration of a miner and his lamp while making repairs or exploring. The oxygen is inhaled by one tube, while through an exhaler is ejected the CO_2 , which is absorbed by caustic soda, leaving the N only to return to the bag. Fleuss' apparatus looks like a knapsack, weighs 28 pounds, contains a 4-hours' supply of oxygen, and has besides a self-contained illuminator—a lamp burning methylated spirit, heats a plug of lime and renders it incandescent, Fig. 110.

The Fleuss diving knapsack, Fig. 110, consists of a cylinder and a cell in four compartments with a perforated false bottom. The cylinder contains oxygen at 240 pounds



FIG. 110.

pressure, and delivers the gas to the nostrils by a tube. The carbonic-acid gas is exhaled by the diver into the cell, where it is absorbed by caustic soda. The entire combination carries a four hours' supply, and has done excellent service to rescuing-parties after accidents arising from fire, inbursts of water or flood of gas.

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PART II.

PRACTICAL MINING

CHAPTER I.

SHAFTS.

61. Shafts: their location, dimensions, and shape; round vs. square; sump and subsidiary shafts; equipment, number, and size of compartments; single and double entry shafts or slopes; shafts for railroad tunnels; mode of sinking, progress, and cost. **62.** Timbering shafts; various modes of cribbing by wood, masonry, and iron; shaft pillars; slope timbering; Hollenback shaft; walling of circular shafts, References.

61. SHAFTS may be sunk for permanent or temporary objects, and they may be intended for one especial purpose only—of hoisting, travelling, or ventilation; or their size may be sufficiently large to warrant division into a number of compartments, one each for the pumping and ladder way, the remainder for hoists, according to the output. Collieries require additional communication with the surface for ventilation. The large area required for, and the foulness of, the return air demand a separate outlet for upcast, as also for the intake, which should never be interfered with by hoisting.

The numerous drawbacks to single-entry compartment shaft or slope are so fully recited in I, **5**, that only in vein-mines should the development be thus risked. Certainly the ventilating ways should not be in adjoining compartments, because the bratticing could never be kept tight enough to prevent a leakage of fresh air into the upcast. Only the expense of sinking in hard, or the difficulties in soft or watery,

ground preclude double entry. Where a prospecting drill-hole has tested the ground the shaft should not be carried down along on it, for the drill-hole will eventually be of greater service as a ventilator and a ropeway than it could be capable of during sinking.

When it is desired to remove the mineral quickly, several shafts are sunk, their positions being a matter of indifference. Ordinarily, however, the location of a shaft and its equipment is a matter of vital import. The configuration or nature of the surface affecting transportation may govern the selection of a site; but, *ceteris paribus*, the principal shaft should be so located as to reach the lowest point of the workings. This is not at the outset always possible to do, so we are accustomed to see one shaft after another abandoned or relegated to secondary uses. Instance the numerous illustrations from the Lake Superior region. The Calumet and Hecla has eight shafts, each over 3000 feet deep, and four of 1000 feet with a complete plant over each one. Nor is it the exceptional case. The prospective depths of shafts are not limited by considerations of a mechanical nature. Hundreds of shafts now exceed 1000 feet in depth where in 1880 there were few.

Shafts sunk to facilitate the execution of long tunnels are best located with their axes in the plane of the tunnel, affording better alignment, and only because of the difficulty of supporting the shafts at the tunnel level is it the common practice of placing them to the side. Shafts are, however, losing their importance for this work, since the introduction of the rapid, ventilating, drilling-machines.

As regards form, the rectangular is the most common (Fig. 111). Its timbering is easily accomplished, and the best adapted to loose ground. Where brick or stone is used instead of wood for lining, the sides are arched to give great strength, and this perhaps led to the round or elliptical shapes, which are such favorites in Europe on account of their greater resistance, and particularly because of the loose soils and watery strata encountered. That their entire area cannot be utilized is, however, an objection (Fig. 112). The timbering of the polygonal (12 to 16 sides), used in Belgium

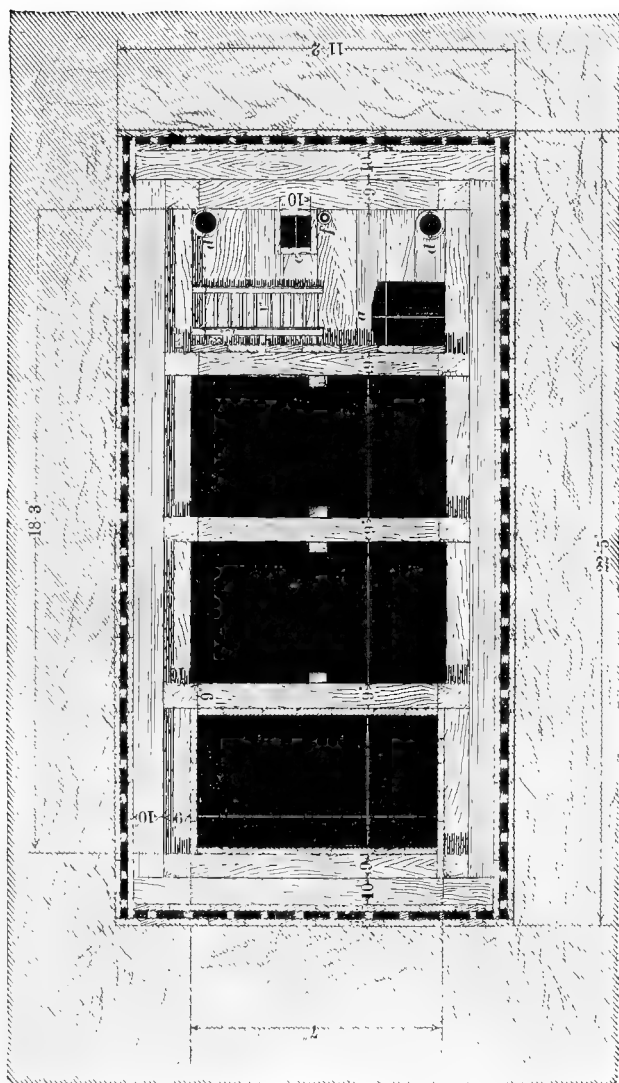


FIG. III.

and the North of France, is not so easy to fit as is that in the hexagonal or octagonal shafts.

The dimensions of the shafts, governed by the number of

compartments, should be carefully studied to meet all requirements of strength, output, and escapement for a prolonged period. The scale increases as the depth and output is large. Outputs of 100 tons were regarded as large not so long ago; but now many hundreds of shafts have a capacity of 1000 tons daily. They are larger on the Continent than in Britain, and colliery shafts demand a greater area than do those in metal mines, which have less traffic, besides being restricted generally

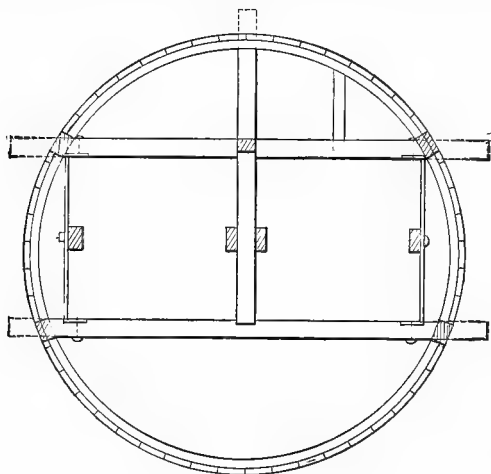


FIG. 112.

by the distance between the walls. The size of the compartment is determined by that of the bucket, skip, or cage, its length being the width of the shaft, the length of which is governed by the number of divisions (see 1, 23 and 27). Compartments placed side by side make a stronger shape than if arranged in a more compact form (Figs. 113 and 114). The



FIG. 113.



FIG. 114.

compartments for metalliferous cars are about 4 ft. $\times$ 5 ft.; those for the coal cars, from 6 ft. to 8 ft. wide, by from 10 ft. to 12 ft. long, measured inside of the timbers. The common sizes for coal shafts are 10 ft. $\times$ 38 ft., and 12 ft. $\times$ 24

ft., with wall plates of some even 50 ft. long. In the Lake Superior iron region the shaft dimensions are about 9 ft. long and 20 ft. wide. In Montana and Nevada smaller sizes prevail, while in Colorado a single compartment suffices for the small outputs of high-grade mineral. The largest shaft yet begun is a nine-compartment shaft 38 ft. $\times$ 42 ft. in the clear. Circular shafts for buckets holding about 1500 lbs. are 8 ft. in diameter; for cages 13 ft. The sizes of the ventilating shafts are a matter of indifference, so that they transmit the necessary volume of air with the minimum resistance, and at a current velocity not exceeding 1000 feet per minute. The upcast shaft is therefore usually round, and the downcast a walled rectangular. Neither should be housed, though the former for a furnace ventilator may be provided with a chimney high enough to prevent the distraction of the current by surrounding buildings; or with traps closing tightly and quickly if a fan is used. An area of one square foot for every eight men employed is a good basis for the upcast of a moderate-sized mine.

The features governing the selection of site have already been examined on p. 20; so there remains to consider the process of sinking. In a soft-ore lode the shaft section should reach from wall to wall, and massive shaft pillars be maintained, else it is sure to succumb. In hard-rock lodes the shaft should preferably be on the foot-wall; on the hanging-wall heavy supports are necessary, especially if the country rock is porphyry.

The sinking of shafts is laborious, because of the difficulty of putting long angling shot-holes. Small shafts are sunk by hand cheaper than by power drills, and almost as expeditious, unless perhaps the continuous system (see No. 91) is used; and the loss of time in removing all the implements for each shot bears a large ratio to the total. Even in drifting, the actual drilling heat is not more than half of the whole time. The number of men depends upon the size of the shaft opened; only two miners can drill to advantage on an area of 20 sq. ft. A larger size gives more room proportionately to each miner, and

permits faster work, and in a shaft 10 ft. $\times$ 11 ft. there is room for three pairs of miners. This space will accommodate two machine-drills, which in ordinary rock can make 5 ft. of advance per day (divided into three shifts of 8 hours each). A shaft long in proportion to its width, sunk by two or four machines, has two centre-cut ranges of holes (see 90), which are independently fired. The cost of sinking is from \$5 to \$18 per cu. yd. Below 100 feet the rate increases each 100 feet almost as the square root of the depth. Rziha says that in Europe the cost of excavating shafts is from 50 to 100 per cent higher in wages, and the cost of putting in timber 15 to 30 per cent higher in wages than the estimate for the same amount of tunnel-work. In the Lake Superior region one lineal foot of average shaft costs as much as a lineal yard of gangway and a cubic fathom (216 cu. ft.) of stoping. There is nothing but a local criterion for the means of calculating the cost of any kind of rock-work.

Through the first score of feet the progress is quite rapid; the dirt is thrown up to the surface from platforms; beyond this, small shafts can be carried quite satisfactorily for 90 feet or so by windlass, but as an engine must ultimately be used, it were better to place it at the start. The entire section is attacked at once, a small corner sump being carried in advance for drainage and for "bearing in" while shooting. Often a hood is provided for the protection of the miners against falling of small rocks, and trap-doors at the surface too, unless the ventilation is poor. If a shaft is to be prolonged while the upper part is still in use, safety is obtained by opening only that portion of the shaft area not under the hoistway for a distance of 12 or 15 feet, and then widening it out to the entire size of the main shaft. This leaves a roof of rock ("pentice"), Fig. 115, that shields the men. When another lift has been sunk, the pentice is cut away, and another started for the next drop. Hoisting is by underground engine or bucket and windlass. A box-pipe, projecting some distance into the air, from over a stove or burning torch, will furnish almost as good air as a

small fan or the air from the power-drills. Except in the neighborhood of oil and gas lines, no especial precautions are necessary against fire-damp.

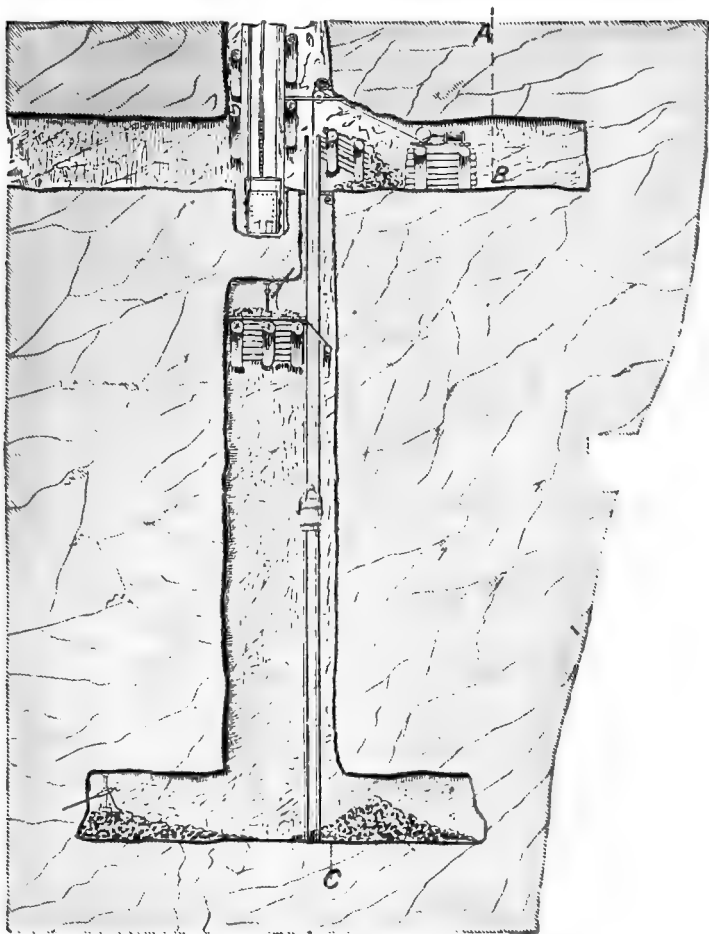


FIG. 115.

62. There is neither safety nor economy in the practice of leaving the shaft untimbered, even if the two walls are hard and self-sustaining, and shoot clean. To resist the thrust of

the country, timbering or lining is urgent (see also notes on hoisting). This may be done simultaneously with or subsequent to the sinking, according to the firmness of the ground. Each timber set is supported on its stulls, resting in notches ("hitches") in the rock; or sections rest on heavy reachers at every 25 or 30 feet of depth. Sometimes the timbers are hung from an upper frame by spiking one set to the other.

The timbers are preferably dressed, though hewn logs are much used for solid crib-work where plenty of help and room is had. Their size is not a matter for calculation, as in firm, non-decomposing ground they experience little pressure, and stability rather than strength is sought, the latter being secured by ample shaft-pillars. Under such conditions a lining with stiff guide-planks is sufficient. This may consist of 3" planks cut with shoulders in sets of four pieces, two wall-plates and two end-pieces (Fig. 116). If cut by template or by

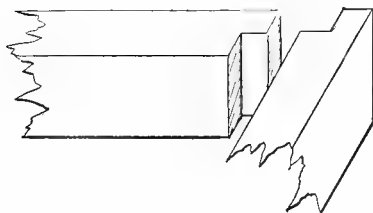


FIG. 116.



FIG. 117.

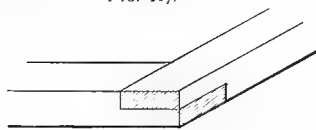


FIG. 118.

machine similar to Fig. 195, they need not be matched in height. This casing is placed in position, lined by plummet at the corners, not spiked or joined, but simply held up to bearings by waste rock packed close between it and the rock. Each 30' section is held on a pair of 10" stulls. Two men can complete one section of a 5' by 9' shaft in four days, with a helper at the packing. The men are supported on a cradle, suspended by a rope from the upper stulls. In very good ground this casing will suffice for three compartments, but not for cage use, unless perfectly backed. In bad ground a casing of larger timber, say 8", is not infrequent (Figs. 94, 117, 118). These are laid "skin to skin," with their ends shouldered.

The wall-plates are stayed by "buntions" (Fig. 119) bolted or gained into them. The longer the wall-plates the stouter the buntions, the interior side of which and of the end-plates carry the cage-guides. The travelling and pumping ways are partitioned off by planks nailed vertically to the buntions. Another method, which is still better, is to have the wall-plates break joint with the end-pieces instead of arranging the four in a horizontal set. The reachers are hitched into the floor and forced down against the hanging-wall. None of these plans are practicable with inclines, which will require framing.

In framing vertical shafts the stulls are inserted into both walls horizontally. On the reachers four sticks are placed and framed to the studdles or struts at the corners and at compartment portions. On these struts a similar set is framed 6' above, to in turn support another parallelopiped, and so on up. Planks ("lagging") are driven in around these frames, and the spaces to the rock filled with broken waste. The joints of each timber are of the pattern shown, Fig. 120. Fig. 121 is

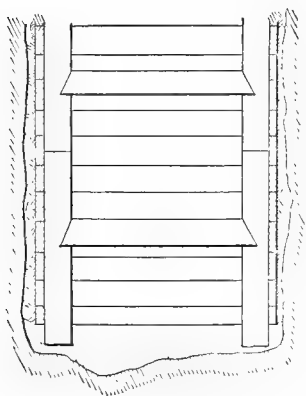


FIG. 119.



FIG. 120.



FIG. 121.

better carpentry, and quite standard. The end-pieces and struts are usually square, 8", while the wall-plates are laid 8" vertically and 10" or 12" horizontally. The buntions are stouter as the wall-plates increase in length (Fig. 122). Fig. 20

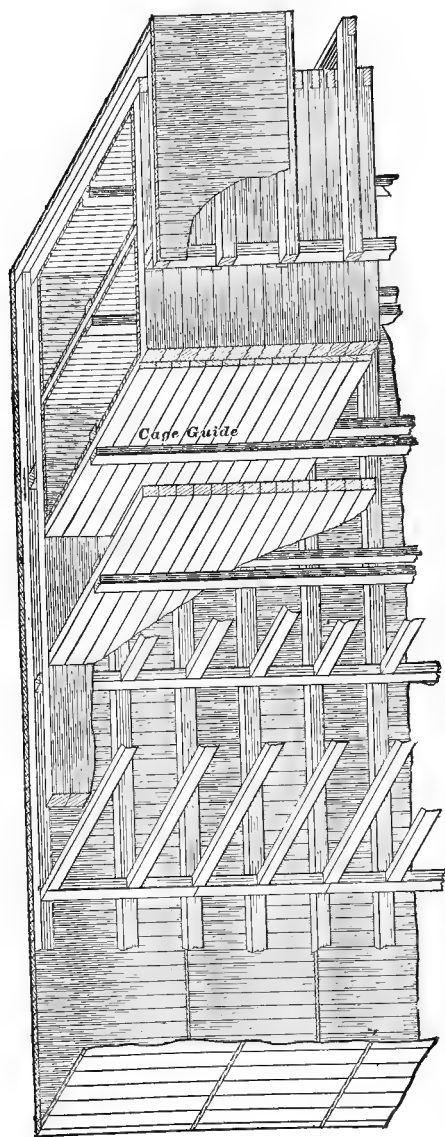


FIG. 122.

illustrates another form of timbering rectangular shafts with vertical corner-plates and horizontal lagging.

Shafts such as the Comstock, 6 by 24, for prolonged, rapid, heavy hoisting are fitted with timber as much as 14" square, and lagged with 3" plank. Where friable rock is penetrated, the frames are braced by inclined struts that prevent settlement. When the ground is friable, marly, or wet the methods approach a caisson character. Another plan comprises a stout framing as described, inside of which is another strong planked cribbing, between which clay is puddled to exclude surface water. The B. & O. shaft at Taylorsville, Ind., was thus successfully carried through quicksand; the outside crib was of 12", the inside of 10", timbers, with a 4" puddled wall. The famous Hollenback shaft, 45' 4" $\times$ 11' 6" inside, has a 12" clay wall for 31 feet deep (Fig. 122). It was designed for a daily output of 2500 tons of coal.

If the timber shows signs of giving way, other means of securing the shaft must be invoked. With expert timber-men the joints may be strengthened or the frame replaced, but it is preferable to reinforce them by sets closer together. Where the expense would warrant it, and the diminished area is not objected to, the insertion of a second lining may secure the works. In the Lake Superior region, after futile experiments with other accessory modes, iron caissons were invoked. In stiff ground they were forced down inside or outside of the old timbers; in soft, they sank by their own weight with the undermining. The cylinders were in segments and sections, bolted at the surface, keeping pace with the progress, averaging a foot a day. A cast-iron cylinder 15' diameter, 1 $\frac{1}{8}$ " thick, was forced down 84 feet at a rate of 2 feet per day in morainal matter.

Forepoling, a form of sheeting (see Fig. 208), is also quite successful, but requires much timber. When the ground is treacherous there is a constant contention against the rising of the bottom. In the event of this happening, the simplest plan is to floor and brace the bottom, advancing the small opening by forepoling (Fig. 104) and subsequently enlarging the shaft to its full dimensions.

For circular shafts the framework descends with the shaft

in sections, which, however, are built upward from reachers, bedded whenever suitable foundation offers; or the "curbs" rest on a properly dressed ledge of the rock, and are firmly wedged against the sides. The timbers, assuming the character of *voussoirs*, are hooped with iron and called "curbs." The timbers composing the curbs may be mere wedge-blocks, or are long enough to form a regular polygon, when they are held by dogs. In ordinary ground the sets are held apart by props, and the solid-packed lining backs them. Otherwise they may be formed into a solid walling, often suspended from a heavy frame at the surface by iron rods. In any event the

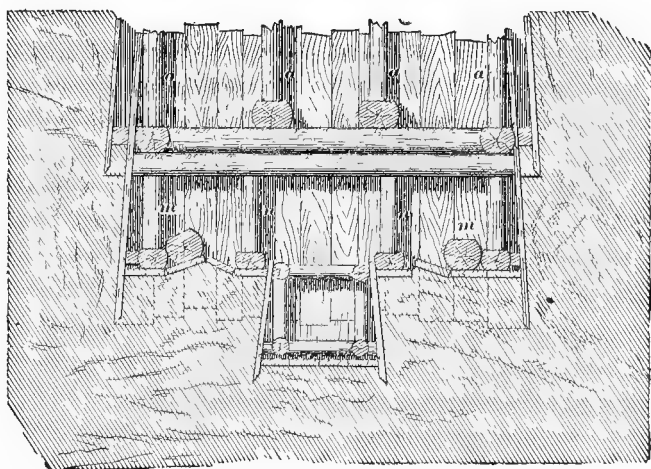


FIG. 123.

joints and fitting receive the greatest care, and many of the old shafts are high types of the carpenter's art.

The increasing scarcity and cost of large timbers, the expense of fitting and maintenance, their short life, and, finally, the corrosion of spikes and splice-plates, with the consequent leakages, have caused the abandonment of wood tubing, and the adoption of iron and masonry for all permanent ways. The effect of the heavy, hot atmosphere of the mine upon timbers is a decomposition, that is not always detected on the

surface, but once begun, only better ventilation can delay ultimate destruction. Dry timbers should be frequently probed; alternations of wet and dry are exceedingly destructive; wet timber will last longer than dry. Preservatives have been attempted, with much success. In salt-mines steeping in brine gives great endurance. The sulphates and chlorides of zinc have proven excellent antiseptics; and a grand opening offers to the discoverer of a means of freeing the lead ores of the Western States of the obnoxious zinc, and at the same time utilizing it as a preservative.

The use of masonry for the walling of shafts involves but one disadvantage: it presupposes ground that will stand safely for a couple of weeks without much support. Before the permanent structure can be introduced, a considerable depth must be reached, to obtain a sure foundation upon reachers, or upon a ledge from which the masonry is erected, the temporary timbering and bracing being gradually removed as the construction proceeds. When a very secure ledge or base cannot be had, a wedge-shaped chamber is built for some distance back into the rock from which the solid crib supports the walling.

If the pressure from the walls is not great, the brick or the lode-rock is built up in plane walls, packed behind by waste. Often the mine water carries matter in solution that cements the whole into one solid mass. When great pressure is expected, the sides are arched toward it; and in very bad ground all four sides are curved, or the circular form is adopted. The arc should be such that its chord is perpendicular to the direction of pressure. In such event, the foundations for the sections are curbs of overlapping timbers patterned to the curve, or of late years of cast-iron, with slabs of wood at the joints. The packing behind is carried up with the brick or masonry until the ledge of the upper section is reached, when it is removed gradually and the two sections united. In some instances the masonry compartments are built at the surface and lowered into place. Brick is well adapted for quick arch-work. The wall of a shaft 13 ft. in diameter is four half-bricks thick;

the labor of laying it from a staging is one and one-half days per M. The masonry is supported by rods, *b* (Fig. 124), from beams *a*, *a*, buried firmly in the walls.

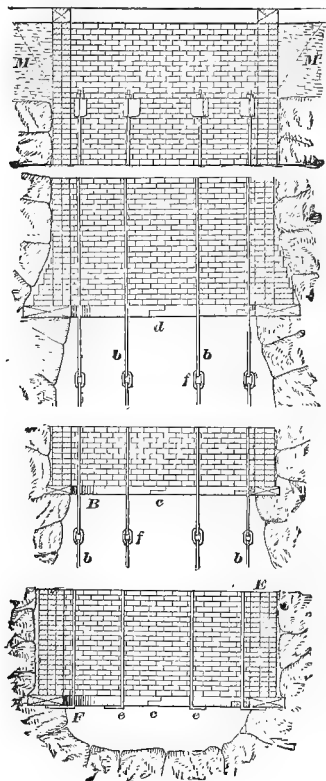


FIG. 124.

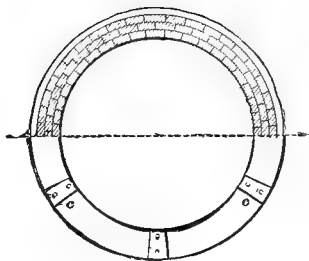
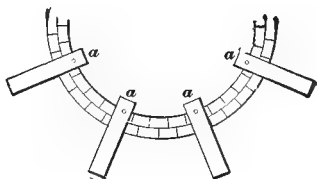


FIG. 125.

Masonry is heavy to support, and not any cheaper now than iron, with which many shafts are successfully curbed. Rings of I beams or channel-bars form the curbs, upheld at proper distances apart, by struts of wood or iron, and backed by heavy planks or $\frac{3}{16}$ sheeting (Fig. 112). English engineers use old railroad iron similarly. Prepared at the surface, the curbs may be lowered into place and quickly set, with little labor. A packing of concrete is used at Saarbruck, giving increased strength and durability. It is estimated that the

initial cost of iron lining in place is twice that of wood and equal that of masonry, but the cost of maintenance is one

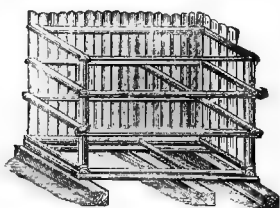


FIG. 126.

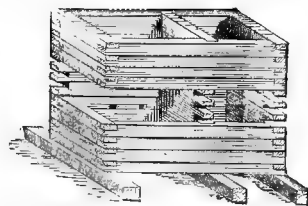


FIG. 127.

third that of wood and nearly the same as with masonry, if the shaft is dry.

Fig. 126 illustrates the mode of timbering a shaft in firm ground by sets of unhewn timbers at 3 to 6 ft. apart, lined with vertical planking.

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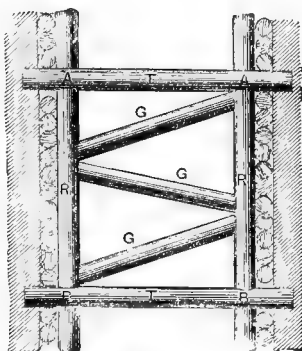


FIG. 128.

CHAPTER II.

SINKING IN RUNNING GROUND.

63. Precautions taken to exclude water; tubing; description of and estimates for Triger's method. 64. Kind and Chaudron process of tubing and sinking through watery strata; description of the tools; estimate of cost; applicability and advantages; examples; Haase's system; J. Mill's Californian method; Poetsch's freezing process. References.

63. WITH an increasing zeal in the search for minerals, mining is conducted to greater depths and into more treacherous ground, as time advances, and the highest type of engineering skill is called into play as water-bearing strata are encountered. Not only is the shaft to be sunk through them, but their underground currents must be excluded from the mine by water-tight casing, otherwise an elaborate system of pumps must be continually maintained during sinking and mining. Not infrequently the expense of removing seepage-water while sinking becomes so large an item of dead-work as to compel the abandonment of the works. A system of tubing is, under those circumstances, advisable.

Two varieties of cases present themselves—one in which the ground penetrated is quite firm but watery, and the other includes running soil, marl, quicksand, etc. In the first case the pumping facilities must be ample, or the water kept back during mining; in the second, the excavation is more rapid than the facilities for removal to the surface permit, and there is the danger of overwhelming the laborers with soil. In either case the ground traversed must be insulated by a tubing, hermetically sealed above as well as below the soft measures.

This not only renders sinking possible, but it excludes water and silt from the mine, and permanently dispenses with much of the pumping arrangements. It very seldom comes into play in vein-mines, where, with the verticality of the lodes, water cannot be prevented from percolating into the mine. It is in the stratified regions that the use of the crib is of the highest importance. Beds of gravel, sand, and clay, or porous strata, percolating large quantities of water, are not easily traversed or held up without a strong water-tight lining, for the pressure of the moving material tends to make the bottom rise, as well as threaten the sides. A deep shaft in such a region may encounter several occasions for such tubs, which under suitable conditions may be introduced in lengths as required, and only to the extent of the soft ground. Still, it would give more substantiality to the work to form one continuous length of tubbing, even across the good ground. It is not uncommon to find in Germany shafts with three sections of iron tubbing, united by lengths of brick or wood lining.

This process of tubbing consists in confining the seepage area to that of the bottom only, by building a water-tight cylinder lining to the shaft, and carrying it down with the sinking beyond the wet stratum. In England, a bed of sand called the Lower Red Sandstone, which is almost fluid, has several shafts tubbed through it. In Britain, Belgium, and the North of France several mines are reached by tubbing through the fissured chalks and marls of the Cretaceous. The Thonmergel of Germany is frequently tubbed to the Bohn Erz, below, dry enough for work. While sinking the Murton pits 4000 gallons were pumped per minute, and the "come in" of water for the Exhall shaft was 1650 gallons. Still, the inefficiency of this plan, sometimes called the English system, is recognized, and several methods better applicable to loose and watery beds have been applied with more or less success. Excepting the Poetsch method of freezing the ground to be penetrated, they are modifications of the diving-bell, or pneumatic pile.

Wrought-iron tubes in segments are bolted on at the surface as fast as the lowering proceeds, until the secure, imper-

meable bed is reached. Here a smooth base is prepared for one or more wedged curbings, behind which moss or concrete is rammed. The tubing is backed with rock or concrete all the way up, and connected with the next upper section. The holes in the segments, for convenience in handling, and to relieve the tubing of pressure till the work is completed, are plugged up. The early practice of bolting the segments together through the inside flanges was soon abandoned, and now the flanges are outside, wedging, pressure, and friction keeping them. On account of the curious accidents occurring from the pressure of air locked behind the tubes, it is advisable to lay a pipe to the surface for the gas to escape, and, similarly, to relieve the water. A shaft of 16' diameter was sunk at a monthly average of 104 feet with four shifts of 8 and 10 men each. Several Canadian salt-mines, having shafts 10' 6" diameter and reached at a depth of 1150 feet, are tubbed through 260' of water-bearing strata, in sections 2 feet high and $\frac{5}{8}$ " to $\frac{13}{16}$ " thick. The columns rested on iron curbs with firm base. The joints are calked.

Tubbing of masonry or of timber, once much employed, is cheaper than iron. With good hoisting-machinery, three masons in four-hour shifts finish 10 feet per day of a 16-foot shaft. Towers of masonry, resting on an iron curb with a cutting edge, were built on at the surface; while, to facilitate the sinking, digging was being carried on below, or, if the material was wet, a process of "bagging" was employed. When abundant in size and quality, wood gives great satisfaction, as being elastic, easily laid and repaired—qualifications not possessed by masonry. Iron offers the advantages of strength, combined with a facility of handling, which recommend it for large shafts and enormous flow. Though it is not possible to presage or measure the pressure, and thus determine the kind and strength of tubs, a thickness, of 12" wood, 7" masonry, and $\frac{7}{16}$ " iron may be suggested as common. As a matter of fact, the tubbing should taper off toward the top. In many cases, however, the use of 12" staves hooped with iron did not prove adequate, nor did the backing of 12" more of concrete help matters; where the shafts were not abandoned, $\frac{5}{16}$ " sheeting met the emergency.

When a bed is encountered of a material so soft as to behave like a fluid and be pumpable, not even as small an area as that of the shaft can be opened with safety: in this event some variation of the "spilling" processes may be employed (71); or, failing of them, the pneumatic pile or some process of congelation should be employed.

M. Triger employed, 1839, the principle of the pneumatic pile, in which the iron tubbing, an indispensable complement to boring through watery strata, extends down to an air-lock communicating with a diving-bell at the bottom. The atmosphere of the caisson is maintained at a pressure of not over 60 lbs. per square inch, and checks the influx of the sand, which the miners shovel to a sump, whence it is aspirated to the surface. Meanwhile the tubbing is being rammed at the surface. In the air-lock, which connects the diving-bell with the surface, the men are prepared for the change in pressure. A windlass in the air-lock and one at the surface raises the coarse stuff in two stages. The physiological difficulties prescribe the applicability of this method to a depth of, not to exceed, 150 feet.

64. Over a very extensive tract of country in France and Germany the loose, watery marls presented difficulties which the methods described failed to overcome. What with pebbles and fine rock interfering with aspiration, water completely inundating the shafts, and the difficulty in establishing water-tight joints, the operators were routed. In 1850 Herr Kind devised a scheme for mechanically sinking shafts, just as one does a bore-hole, and still further conquered difficulties hitherto insurmountable by a variation in the mode of lowering the tubbing, and by a device for regulating the influx of water. When M. Chaudron added the sliding bottom-piece to form a perfect joint, after the Kind boring-tool had prepared the base, the acme of shaft-sinking was reached. Since 1862, when the first shaft was sunk, 6 feet wide, 480 feet deep, at a cost of \$450 per foot, not a single fatal accident is recorded against the process, which owes much of its success to the fact that the sinking and lining are completed before a soul enters the shaft. Two abandoned shafts, through soil feeding 11,000

gallons per minute, were carried down 267 and 216 feet, respectively, in 23 and 20 months, with a cost of \$280 and \$340 per foot. In the latest application 569 feet were sunk, 16 feet diameter, at an average cost of \$143 per foot for both shaft and lining, which alone cost \$70 per foot. With a guaranteed success at so low a rate, it is surprising that American engineers, usually so progressive, have not employed this method before acknowledging failures; but no attempt to introduce this plan here is as yet recorded.

The mode of procedure consists in first drilling a small hole about 50 feet deep and 4.5 feet across, then widening the shaft by a reamer to the required diameter, and proceeding with the smaller drill for another 50 feet. The drill, called a trepan, is operated from a walking-beam by a surface engine.

The small trepan (Fig. 129) consists of a blade of forged iron, into the lower side of which are keyed a number of pointed steel teeth, and a stem connecting the blade to the suspension-rods by means of a sliding-box. This last partially corresponds to the "jars" (Fig. 213) of oil-well outfits, takes up the jar, and is an essential element of the tool. The trepans are massive;—for hard rock, weigh from 8 tons up,—and are raised 6" or so turned slightly for each blow, and dropped; their concussion disintegrates the rock along a diameter of the circle. The progress is from 3" in flint to 3 feet in chalk per day; 1 foot in sandstone and 16" in coal measures.

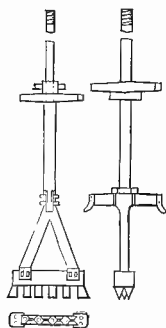


FIG. 129

Most of the material is broken quite fine, though 2" and 3' stuff is not unusual. When the hole has advanced some distance a larger trepan is attached (Fig. 130). This is similar to the smaller one, but the blade is deeper at the centre than at the ends, so that its teeth cut a base sloping to the centre. The central, toothless portion of the tool has a U-shaped guide that fits the smaller hole. This tool, often weighing as much as 15 tons, cuts the shaft to full width, or it may be succeeded by another similar reamer, the detritus falling into the

smaller hole, from which it is hoisted by the sludger (Fig. 132). In alternate stages the drilling and widening progresses, while the tubing is subsequently lowered by a separate engine. All these operations being conducted under water, the trepan

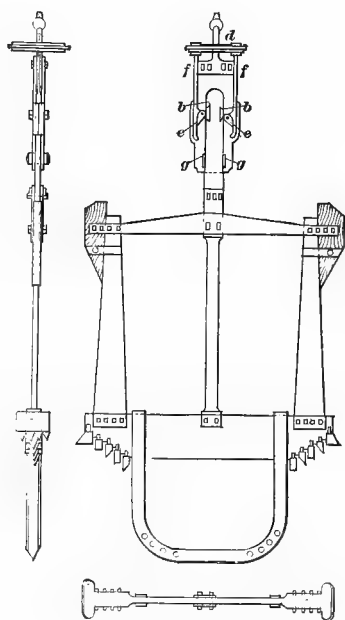


FIG. 130.

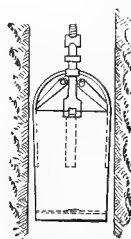


FIG. 132.



FIG. 133.

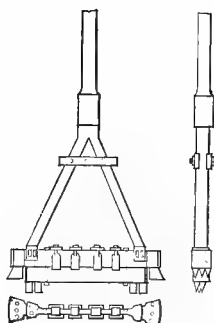


FIG. 131.

requires to be automatically kept vertical. Two guides, carrying at the extremities horizontal and vertical cutters, accomplish this marvellously well. A record of the preliminary shaft, $4\frac{1}{2}$ feet in diameter, showed for 508 feet an average progress of 3.3 feet per 24 hours, divided up as follows: 51 per cent of the time was occupied in drilling, 19 per cent in raising and lowering the tools, 20 per cent in dredging, and 10 per cent in repairs and delays. Widening the shaft to 14 feet and down 460 feet took ten months; reaming, 46 per cent of the time; altering and operating the tools, etc., 14 per cent; dredging, 22 per cent; delays and repairs, 18 per cent.

The small trepan is illustrated in Fig. 129, a widening

trepan with double blade in Fig. 132, a reamer in Fig. 130, and the sand bucket-dredger in Fig. 133.

There are no screws or nuts to loose; all the parts are keyed to place; but special tools are provided for grappling broken rods, stems, trepans, teeth, etc.

At the surface, the operations of boring the pits, building and lowering the tubing, puddling and sealing the base, are conducted with engines and capstans from a tall derrick, at which extra lengths of rods may be attached with the progress of the drilling. Nine men are employed about the works, only three of whom are skilled laborers. The cost of the installation of the machines, tools, etc., all of which are portable, is about \$13,000 to \$20,000.

The tubing, which is indispensable to operations of this character, is of iron sheeting, built on in 6-foot sections, with leaded joints, and suspended by rods (34). 112). The flanges come on the inside of the tubing *bb*, leaving a perfectly smooth exterior, the joints true and bolted together. Two sections

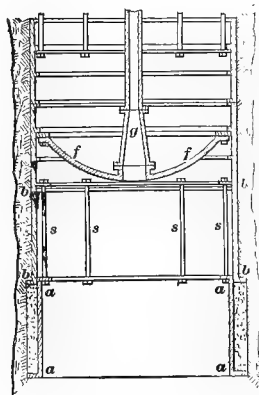


FIG. 134.

are lowered daily. As an example is quoted a tubing 12' 7" internal diameter, 280 feet high; it was 1" thick at the top, $1\frac{5}{8}$ " at the bottom, and weighed 400 tons. The sections were 5' high, the flanges $3\frac{1}{2}$ " wide, 2" thick, having leaden wedges between, $4\frac{7}{8}$ " wide and $\frac{1}{8}$ " thick, and 20 bolts $1\frac{3}{16}$ " diameter.

At the bottom of the iron cylinder are attached two very ingenious appliances, which, operating automatically, have established the process as a success beyond all cavil; the first is a moss-box *a*, for hermetically sealing the lower end of the tubing against any influx of water; and the second is the introduction of a false bottom, *f*, by which the sinking of the tubing is cleverly controlled. These are both adapted to the bottom of the tubing, as is illustrated in Fig. 134. All the flanges of the tubing turn inward except the

lower one of the bottom section, *bb*, which is outside, and may act as an annular piston to a lower section, *aa*, of smaller diameter, the upper flange of which turns inward, and the lower one outward. Between these flanges, the moss-box, and the rock, the annular space is filled with moss, which is not, however, under compression so long as the screw-bolts *ss* support it from the tubing. It operates like the seed-bag of oil-well diggings.

The false bottom *ff* is attached to the tubing, with the lower sections of which it forms a diving-bell, that floats the whole system. The greater the head of water encountered, the more complete the balance, and the greater is the relief to the rods *dd*, supporting the hundreds of tons of iron. The safety-pipe *g*, with cocks and plugs operated from above, is an equilibrium column that permits sinking, or rather regulates its speed. Opened at the top, sinking proceeds rapidly, as the compressed air and water find vent; closed, the whole structure is upheld against gravity. When the plugs are opened they discharge into the tubing, weight it with water, and at the same time release the pressure below. By proper manipulation, therefore, perfect control is had over the lowering of the casing.

When the tubing has traversed the water-bearing measures, a seat having been scraped for the moss-box, the entire weight is allowed to fall on the annular piston *b*, by opening *g* at the surface. The moss is compressed to a hard, water-tight mass, the rods *s* gliding in their bearings (Fig. 135). Up to this time the shaft is more or less full of water (the process is independent of the amount of water encountered), which may now be pumped, but usually is not until a cement backing has been inserted and hardened to insure solidity; after that, if the joints of the metallic column are well made, the shaft is perfectly tight, and the mine is insulated from the subterranean current. The introduction of the cement is effected by a closed spoon holding a barrel or so, curved to suit the space. Three sets of six men each do this work, burying 400 cu. ft. per day, at a cost of

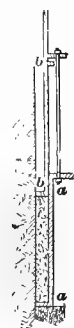


FIG. 135.

about forty cents per square foot area of lining. A solid foundation of wedged iron curbing is subsequently built on a stout ledge, to take the weight of the cribbing after the other work has been completed.

This method is generally applicable to conditions of soft ground, and especially in watery ground, which can be pierced without recourse to ponderous pumping machinery. Though the pressure of the water is not essential to success, it materially facilitates operations. In a few cases, where the ground was merely wet, not running, tubbing was cheaper than this method. But the facts should not be lost sight of, that none of the delays, perils and discomfitures of the ordinary methods are here experienced. Its progress is greater, and initially its maintenance is cheaper than other schemes for wet ground, besides never having had a failure, though no shaft of over 14 feet in diameter has yet been sunk by this method.

Herr Lippmans is using a drill of a double V-shape, instead of a straight trepan. It does faster work, as it cuts equally at the periphery with the centre of the circle. With Kind's trepan the blows fell too far apart at the outer edge of the shaft, and too near together at the centre.

An objection to the Kind and Chaudron method is, that there are no means of knowing when the water-bearing stratum has been penetrated and when the tubbing should cease, unless the preliminary geological examination has revealed it.

A short while ago a pair of shafts were sunk in Samlund, Eastern Prussia, for amber, through 147 feet of clay and sand, by a variation of this method. The drill-tools, weighing 1700 lbs., cut a 4' 6" space, though they had little to do except in the shale-beds. No moss was necessary, as the ground was not wet. Four-foot lengths of tubbing were forced down by jack-screws, each shift with 27 men. The total weight of tubbing in each shaft was 45 tons, and the total cost \$17,500.

Haase's system is a modification of sheeting-piles,—small round iron cylinders driven close together to form a cribbing for the intended shaft. The tubes were about 15 feet long, $\frac{1}{4}$ " thick, and 4" diameter, and enclosed an area 10×7 , which was then timbered. These were driven through 90 feet of quick-

sand in five months' time, at a cost of \$135 per foot. While standing, they gave good drainage, and did not yield when the excavation of the shaft began.

For dealing with loose saturated alluvium, Mr. C. Poetsch has originated the novel idea of freezing the mass to a solid by boring a concentric series of holes through the alluvium about three feet apart, lined with copper tubes, inside which are smaller tubes. Two such rows completely surround the shaft. A very concentrated solution of the chlorides of magnesium and calcium circulates through the tubes and freezes the ground, after which the pit can be excavated in the centre of the mass in the ordinary manner, and the tubbing put in. This refrigeration is continued till solid rock is reached. A shaft 7 ft. $\times$ 11 ft. was sunk through 26 ft. of quicksand, the frozen wall enveloping it being 7 ft. thick, at a cost of \$190 per foot.

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Coll. Eng.: Triger's Method of Sinking, XV. 18; Water Dams in Shafts, XV. 42; Freezing Water an Aid to Mining, XV. 86; Freezing Process, Gobert, XVII. 171.

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CHAPTER III.

TIMBERING.

65. The use and preservation of timbers; for jointy rock, horses, and disintegrating rock; consumption of timbers in mines; selection of timbers. 66. Props, sprags, stulls, and their plates; formulæ for strength and the calculation of their dimensions; variety of joints. 67. The construction of setts, frames, etc., for various conditions of roof, walls, etc.; timbering for levels, gangways, gob-roads, and for support of vein, gangue, etc.; in salt mines; lagging; wood, iron, and masonry for levels. 68. Square setts, joints, and sizes of parts; full account of the American method; cribs for rooms; timbering of mill-holes, underground chambers, plats, and winzes; timber-man's tools; framing-machines. References.

65. WHEN one examines the story that the accident-tables of page 306 tell, it becomes manifest how the neglect of a few simple rules endangers life and property; and in no respect is this more painfully impressed than by the mortality record of unpropped rock. Excavations, even in the "rock of ages," cannot be left open any great length of time without support, which, if introduced in time, will prevent disastrous results. Successful superintendents personally watch the timbering and the face-rock diligently, and guard against any springing of the walls. All the effects of pressure are intensified by neglect, and the secret of success is to place timbers before movement begins. Supports are not for bad roofs only; while "awaiting a weak spot, the good roof, so called, catches him," and his stope or room is lost. The eagerness to quickly win the face, while pardonable, promotes avarice, parsimony, want, and then provokes collapse.

Though the conditions underground are such that very

simple timbering is required compared with that on the surface, the tendency of the time is toward the employment of special timber men to make and place the supports. In rooms, driving in soft ground, and the like, the miners must at once prop the excavations. For this reason—and the proscribed space—the character of the timbering should be simple and the sticks light. Fortunately, the pressure of the country rock is inward and toward the openings, and compression, not tension, as on the surface, is to be combated with. This tends to hold the sets together. Tenons and framing may therefore be dispensed with, except in loose ground, where they are beneficial.

The relative merits of the different varieties of wood need not be discussed here. Oak is undoubtedly the most preferable, but the mines take what can be had in the vicinity. Above "timber-line" we are content with "scrubs." Sawn timber is better than hewn, on account of its better resistance to decay; and durability is of importance prime to strength. Again, green-wood is heavy; the ordinary 10" stick, say 6' long, is as much as three men can well handle. Lightness is an essential feature in this most onerous of underground work.

The life of timber varies with the conditions of the atmosphere and care in dressing. It is rarely as great as that of railroad-ties (twelve years). In many mines head-pieces crumble after two years' standing. Wood rots faster, and shows it less on the surface in dry, vitiated air than in moist air. Alternations of temperature or moisture are very destructive. A cotton-fungus mould is a sure indication of bad air, and, being contagious, requires attention at once.

The decay results from the fermenting of the albuminoids of the sap, the admission of water, and the attack of insects, to which several causes contribute,—bad air, damp air, standing water, and oxidation,—causes all of which are mitigated by an active circulation, and materially remedied by saturation of the pores with some antiseptic. Creosote, Kyanizing, or Burnetizing will give greater life to timber. The timber is placed in a wrought-iron cylinder through end doors, after closing

which the air is exhausted and creosote forced in. Pine absorbs from 10 to 12 lbs. of oil per cubic foot, and the hard woods less. The pressure during the operation is 100 lbs. per square inch.

It is this destructibility and the increasing scarcity of supply that is bringing about the employment of iron and masonry as the certainties of future support. Fortunately, the metal-mines above "timber-line" require but a moderate supply. In others, however, the consumption is alarming. The annual estimated timber construction in anthracite mines is 1 cubic foot per ton product; in the L. S. copper-mines, $1\frac{3}{4}$; in Leadville, and L. S. iron-mines, 3; and in Nevada, $4\frac{1}{2}$.

Every 100 cubic feet of coal extracted consumes 3.4 cubic feet of timber; every ton of excavation in running ground requires about 5 cubic feet to support the balance. In the Anaconda mines, Mont., 80,000 cubic feet of timber are used daily; 2000 of such mines would consume the entire forest-area growth. The West Vulcan iron-mines, in L. S., annually consume 2,000,000 feet of lumber and 60,000 pieces of lagging, at a cost of 37 cents per ton of ore mined. In the copper-mines of L. S. this item amounts to from 15 to 31 cents per ton of rock hoisted. So important in the economy of mining and to the safety of lives, the selection and placing of timbers should therefore receive skilled attention; adequate ventilation is equally urgent for their preservation.

The most important question is as to whether the timbering is to be done in a substantial manner at once, or to be considered as provisional. This is only answered according to the importance of the gangway. Rooms and stopes are only temporary, and treated as such. The excavations may be filled with waste or timbered up during the period of their mining. Gangways may or may not be subsequently cribbed or masonried; pump and machine rooms and stables are very substantially lined.

In any event, each piece should be placed conformably to the principles of the strength of materials, and employed in such a manner that its resistance to crushing, rather than its resistance to bending, be brought into play. Joints should

bear the pressure uniformly and their planes be perpendicular to its direction. Sticks should be placed in the line of the pressure wherever practicable, or in such manner as to act like or take part of an arch; then, when any movement takes place, its effect will be to tighten the timber in place. Every mortise or joint impairs the strength of the stick.

66. Single sticks are much used as props to hold the roof back of the men. In long-wall working a large number is used, resting on the floor or on a plate, and hammered into place with a wedge-plate at the roof. They are 6" or 8" diameter, and stand 3' apart, in two or three rows, beyond which the roof caves in on the gob. They remain only a few days, are removed by rows to let the roof cave, and are replaced nearer the face (Fig. 8). An average of 70 per cent are

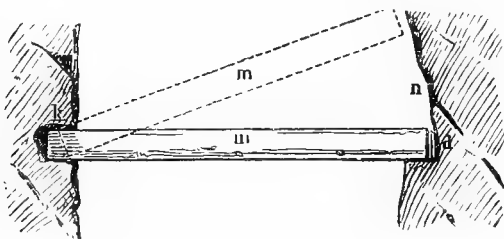
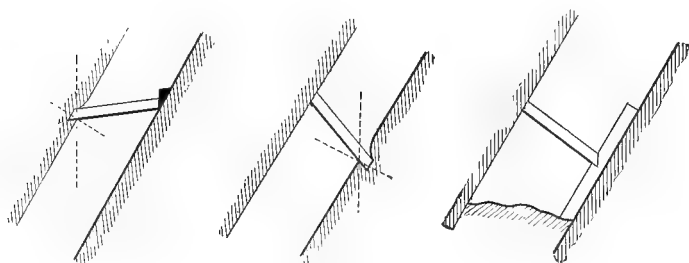


FIG. 136.

recovered; some of the balance cannot be removed; others would endanger the timber-men. Flat caps on top, 20" $\times$ 10" $\times$ 2", are ample for most bad roofs. Slate requires a large plate. It is a poor roof, because it crumbles from the presence of pyrites; sandstone or conglomerate makes a good roof; soapstone is bad; but the most dangerous is fire-clay, which melts upon contact with the air. Props come into play (Fig. 8), 12" long, 3" or 4" in diameter, for holding up holed coal. The props have to support not all the strata above the coal,—this the pillars do,—but a portion of the immediately overlying seams which constitute the roof; and its condition determines the number of props, which is greater for a brittle stratum, and less for a flexible roof.

States vary in their statutory requirements as to the nearness of the props to the face, but 15' is the farthest allowed in any coal region. A distance of 5' or 6' allows ample mining-space.

In metal-mines the prop is used as a stull (Figs. 136, 137,



FIGS. 137, 138, 139.

138), resting in a notch ("hitch"), generally on the foot-wall, unless the hanging-wall is much softer, and driven into place with a wedge-piece, by mallets. In veins of small inclination the stulls are normal; otherwise they set between a normal and the vertical. They are round or dressed, and of a size and distance apart dependent upon the weight of waste stope-rock to be upheld. It is better to increase the number than the

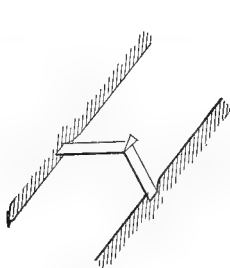


FIG. 140.

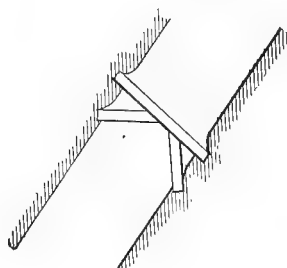


FIG. 141.

size, though their strength is directly as the cube of their diameter. $d^3 = 0.05hw^2m$ is the formula for calculating the size of any round stull held at two ends. h is the height of rock along the vein; m the distance between the stulls; w the width of the vein; d the diameter of the stull. For 60' of stull

dirt, the timbers, 7' long, 30'' apart, are 17'' through, neglecting the friction of the mass on the wall.

If either wall is soft, a broad slab or post laid against it takes the thrust (Fig. 139).



FIGS. 142, 143.

Should the distance between the walls be too great for a single convenient-sized stull-piece to be used, the use of the smaller sticks, in a manner indicated in Fig. 140, is common. A wedge or plank is braced against the walls and extends longitudinally with the drift to be covered. This is not so good as

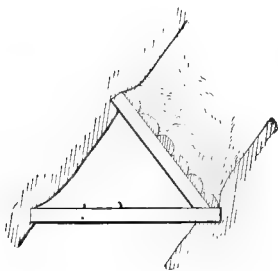


FIG. 144.

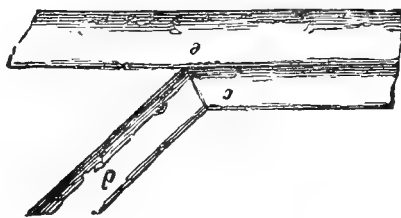


FIG. 145.

Figs. 141 and 142, wherein one or two struts relieve the stull; or Figs. 143 and 144, for underground work for the floor. In Fig. 145 *c* acts like a straining-beam to *d*. Not infrequently the caps may be supported by struts in flat seams, like Fig. 146, or a single centre-prop in double-track gangways and slopes (Figs. 147 and 148); but besides taking room, they are the cause of too many accidents. In fact, much de-

depends upon the cleverness of the men in setting the timbers to the best advantage. For example, a curved stick is bene-

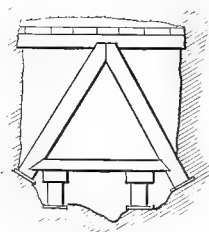


FIG. 146.

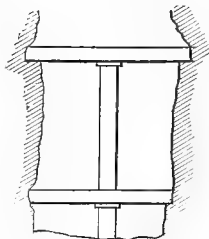


FIG. 147

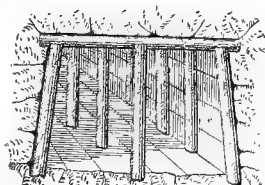


FIG. 148.

ficially placed if used as shown in Fig. 149. It then becomes an arch. The temptation to cut and notch and spike should be restrained, or the tenacity of the fibres will be destroyed. The use of wedges should be avoided where possible, for the weight may fall on one corner and detract from the strength of the post.

The presence of seams and cleavages traversing one another in the rock materially affects the selection of the modes of



FIG. 149.

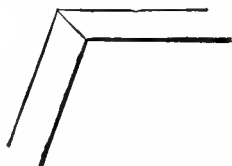


FIG. 150.

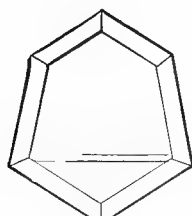


FIG. 151.

timbering. The parallel joints are not troublesome. Horses in the vein usually require attention, as do evidences in coal seams of *sigillariæ*. The latter occur like truncated cones, base down, and the circular layer in the roof should be propped as soon as observed.

In the cases thus far illustrated, the object aimed at was

simply the support of the roof or of the stull-dirt—vertical pressure only. When the stress is also from the sides, frames are made in sets composed of a cap or collar resting on two posts or legs studded on a sill or sleeper. The trapezoidal form is stronger than the rectangular, and is equally serviceable for car-way. This form is susceptible of many varied modifications of shape, frame, and joints.

The only joints desirable from every point of view are the flush or butt (Figs. 150 and 151), provided they are cut with precision. Whether the sticks be round or square, the joints should be flat. Never should a round cap be made to rest in the hollow of the post (Fig. 152), for the fit cannot be made perfect nor the splitting of the post prevented. The cap should be shouldered to bear flat on the leg (Figs. 153 and 154).



FIG. 152.



FIG. 153.



FIG. 154.

When the cap receives vertical pressure only, its entire width bears on the legs, as in Figs. 155 and 156; if the pressure is partly from the sides, the joint is dressed to (Fig. 157); for Fig. 158 the lagging and backing must be firm. The prop and col-

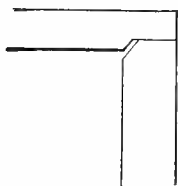


FIG. 155.

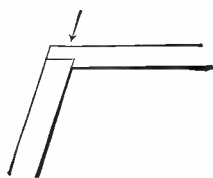


FIG. 156.

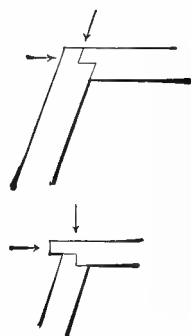


FIG. 157.

lar joint (Figs. 159, 160, 161, and 162) is simple and effective; the bevel-joint is not uncommon in mining-work (Fig. 163);

Fig. 205 is an elaboration of it, seen in large tunnel-work, but is a very injudicious concentration of pressure at one point, the avoidance of which is the very design of framing. The mortise and tenon is very rare in underground work, except in pump-rooms and the like. So, also, there is little use for the scarf-joints, unless perhaps in building beams, for arch centres. Wedges and head-blocks are essentials in the tightening of

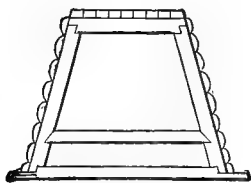


FIG. 158.

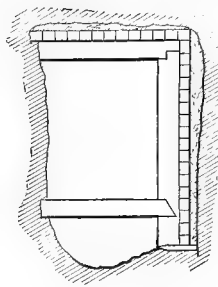


FIG. 159.

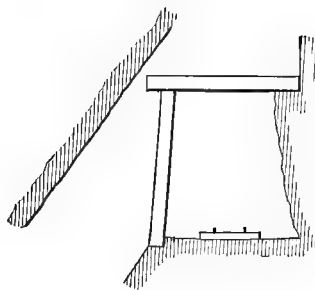


FIG. 160.

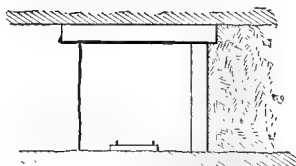


FIG. 161.

frames and to lengthen timbers. Their removal eases the work of reclaiming the sticks whole. The joints ought to be tarred for effective preservation.

Except as clamps, dogs, staples, bands, and spikes, little iron is used in underground work—perhaps 1 lb. for every 100 cu. ft. of lumber placed. Timbers near blasting are often fastened, and bands are used around pieces tending to split. Iron props and screw-jacks may be advantageously employed; otherwise the use of iron is not commended, except as auxiliary

fasteners, for centres, etc. Iron rusts rapidly when in contact with ligneous matter.

The dimensions of the sets are a clear height of 5' 6" or 6' 6", and a width dependent upon the number of compartments. A single way is about 4 feet, though this leaves little spare room. The least dimension of a **heading in which miners can work conveniently** is 3' wide and 4' high. A width

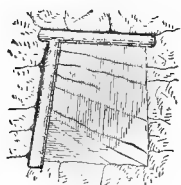


FIG. 162.

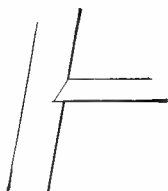


FIG. 163.

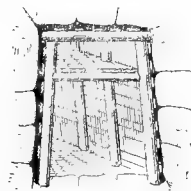


FIG. 164.



FIG. 165.

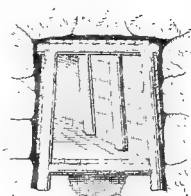


FIG. 166.

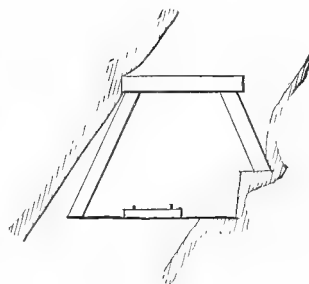


FIG. 167.

of 5' at the bottom, and at the top of 4', is ample for all purposes of a single way. A double way should be 9' wide for men and cars, and 7' high; and a three-compartment way is not much wider, because no provision is made in the haulage-ways for the men who travel in the third compartment. Attention to table on page 222, where it will be seen that fully 12 per cent of the mine fatalities are from crushing by cars in haulage-ways, urges ample room for passing cars, and numerous niches for retreat.

Though the total cost per lineal foot may be somewhat

greater for a wider drift, than for one smaller, the cost per cubic yard of broken rock is less, and the difference in the cost of timbering is slight, but the gain in rapidity markedly favors large tunnels. Compare the quoted results, Rziha's table, No. 70.

Careful observation in the Hoosac Tunnel showed that in a heading 16' wide a man drilled 42'' of holes, removed 0.22 cu. yd. of rock, and advanced 0.058 lineal foot per day, when he could only do 30.119'', 0.134, and 0.047, respectively, in an 11' heading.

In the standard coal-seams a great height is not admitted, though the roof is not infrequently ripped to secure mule height. When the gangway serves as a ventilator or gutter more height is required, and the timbering for the purpose is illustrated in Figs. 164, 165, and 166. The dimensions of the sticks vary, though 14'' is the average used. Large timbers are often needed, but not placed because inconvenient to handle. Instead of building of thicker pieces, the sets are placed nearer than the average—3' 6''; skin to skin is not unusual in shattered ground. The height of slopes depends upon the mode of haulage. The use of carriage requires great height; with a dip of less than 40°, the height is about 7'. For skips, the ordinary height of 6' will do.

Though four pieces constitute the frame, the sill may be dispensed with where the floor is not bad. In slopes they are essential, being wedged into place to secure the trackway. In laying the sills the trench is dug lower in the centre than at the ends, and they will not break. Near their ends the posts rest to support a cap, which is wedged into bearing.

Where a flat road is driven through firm material, the roof of which only needs support, the post and collar form of timbering (Fig. 161 or 162) will suffice. In pitching seams a variety is employed, as Fig. 160, when the vein and country rock are sound and the hanging-wall soft. With a bad roof and good vein one leg is floored (Fig. 167); the other, sometimes longer, rests on the vein. This is also seen in coal-regions. Fig. 168 is a strong form for pressure from sides and top. Again, instead of long sticks in sets, short timbers may be utilized in

framing arches. Sets may be laid close together, or the distance between them may be lagged.

Should the vein matter be too loose to stand up, the gangway is lagged, as is shown, against a long brace (Fig. 168). In wide, soft veins a similar idea employs the elementary form of strength, as in Fig. 169. Gob roads are timbered only at the

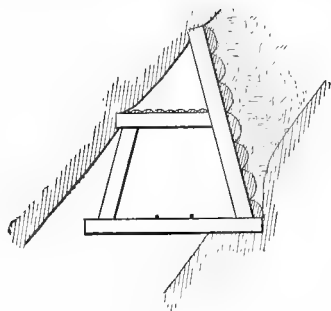


FIG. 168.

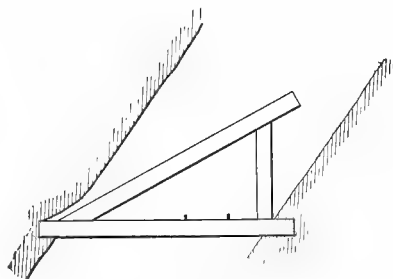


FIG. 169.

roof, by the lagging of caps, resting on dry pack-walls. Occasional chocks will give greater consistency to the whole (Fig. 170). But as the subsidence of the roof cannot be prevented,

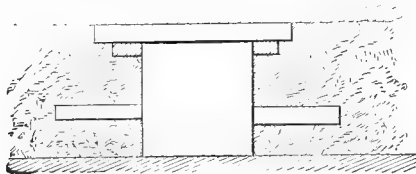


FIG. 170.

the road cannot be kept open long at a time until the gob is pressed solid. Meanwhile the maintenance is a serious item. The plan of keeping a road open along the face of coal through waste is worse yet. The mode of timbering alluvial gold-mines, called "blocking," is said to be perfectly safe by the inspector of mines at Sandhurst. The prop and collar system is used while passing the chain pillar on either side of the gangway; beyond, a modification, in which each cap is, in turn,

made to rest upon the caps of the preceding sets, already built, and upon the collar of the next. The timbers are 8" diameter.

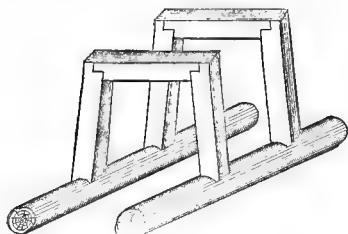


FIG. 171.

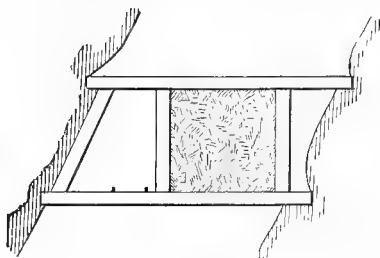


FIG. 172

In very soft ground each cap has two posts for its support.

In timbering wide galleries a line of props form the compartment, like buntons in shaft-work. They relieve the caps of pressure, and are much smaller than the other members of the set, Figs. 147 and 148.

Large lodes are difficult to timber. The gangway takes only a small portion of the width of the vein; the rest, if firm, is left to stand, or if loose, packed with waste. In the Great Devon Consols mine the compartments are used, one for "attle" (waste), and the other for travelling (Fig. 172). The vein runs 22' wide; the stulls were 20" and 18". In a 24' vein a sole-piece of 24" timber and 18" struts, with a longitudinal piece at the apex, made a very strong frame (Fig. 169), which with 3" plank lagging carried from 10 to 50 fathoms of attle. The hitches were cut 18" in. In Southern France, with great pressure from the roof and for supporting heavy waste, Figs. 173

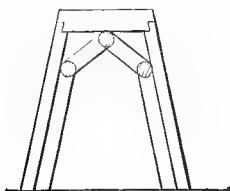


FIG. 173.

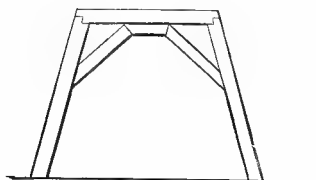


FIG. 174.

and 174 are much seen. In many cases an arch of vein matter 10 feet thick remains untouched from the stope below. With a

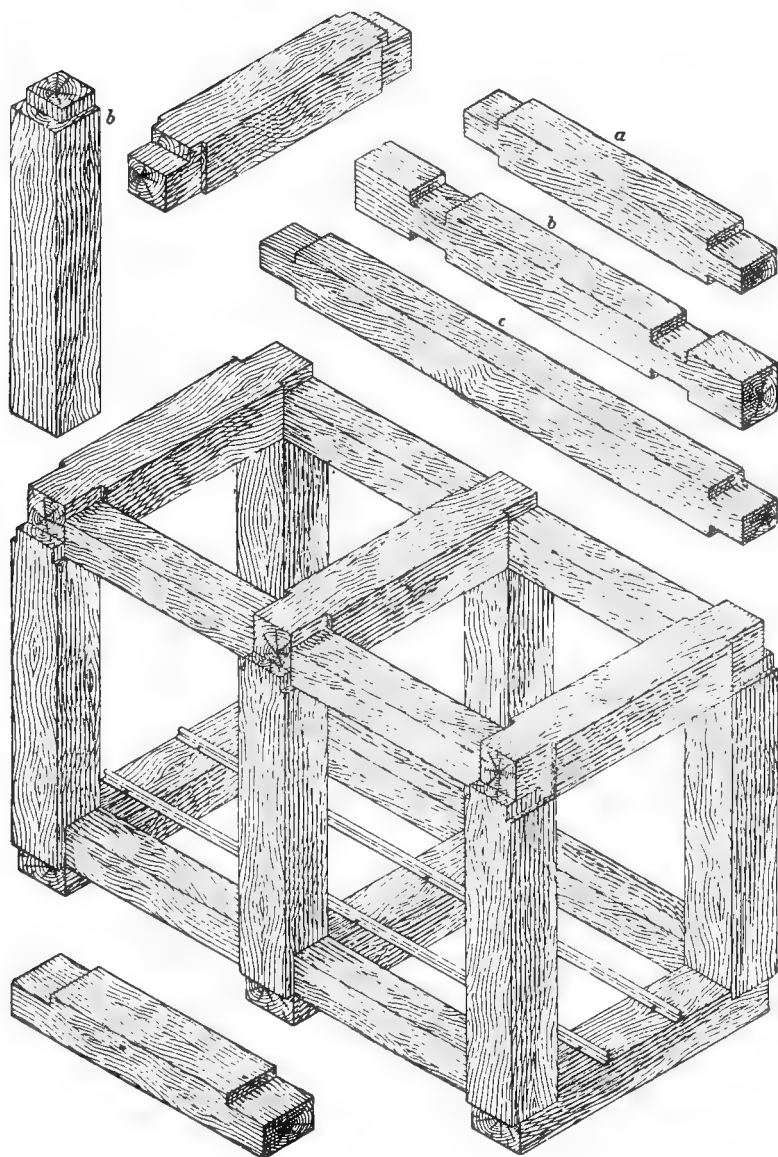


FIG. 175.

system of filling this arch is subsequently recovered. In fact, without filling, no large deep mine can be held by timbers. If, besides, the vein matter is soft, the character of the framing must be entirely altered. In the Austrian salt-mines the problem is very difficult, because the material assumes the nature of a fluid. Also, in rock that decomposes upon exposure to the air the timbering tends to crush, unless an elaborate form of framing is adopted. The gangways in lode or bed may then be a component part of the square-set system (Fig. 175). Where

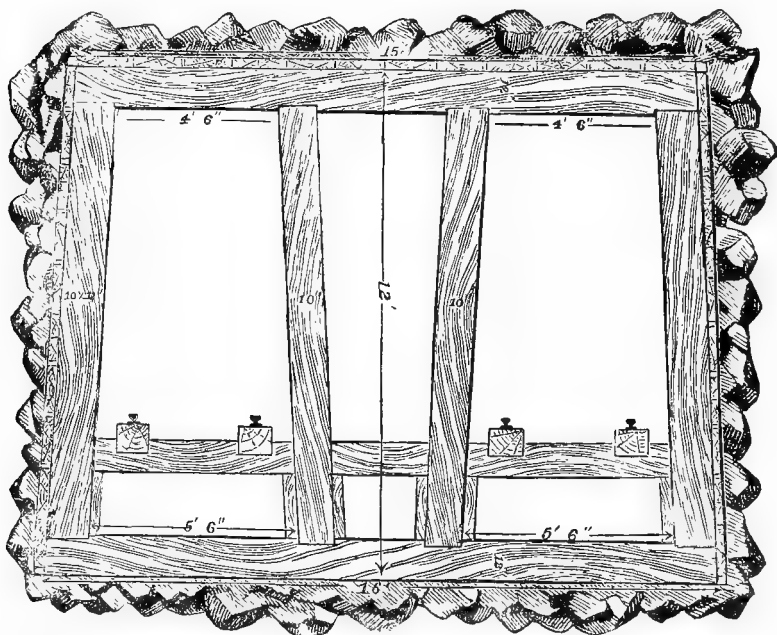


FIG. 176.

ground has a tendency to swell, the only way to save the timbers is to ease up the ground behind the timbers from time to time until the ground settles to its natural state. The swelling can neither be prevented nor resisted. The accompanying cut (Fig. 176) represents the style of Sutro-tunnel timbering, preferable to that illustrated in Fig. 203, which is insecure in shifting ground. Another admirable plan also employed in driving

tunnels is to put a lateral tunnel, or two, very heavily timbered, with an open face to serve as a safety-valve.

Few cross-cut tunnels require timbering further than 50 feet or so from the mouth, for the ground stands well. In pitching rock and porphyry, the roof should be heavily braced.

67. If the spaces between the sets cannot be left open, then the sides and top are lagged with plank or "slabs" from the saw-mills. These are driven close—more frequently flat side to; but reversed they are better. Lagging should never be very strong; the slabs are always weaker than the members of the frames, and serve merely to prevent the fine rock from sifting into the gallery and leaving an open space that gives opportunity for movement, which if once begun can never be resisted. A very slight movement produces sufficient pressure to break the lagging, and thus relieve the costlier work. These are readily replaced on occasion. In many cases round poles are used, 3" or 4" in diameter, overlapping two sets. If the roof does not run, a few logs suffice (Fig. 164). In soft ground or under poor roof the men are protected, and advance made by "poling" ahead of the face (Fig. 204). The open space between the lagging and the rock is packed with waste or wedged perfectly. Brush piled back of the lagging holds up the "smalls" well.

When the capping over the firm ore of a flat-bed is not good, timbering may be saved by not stripping the entire height of the vein, but leaving a layer of it for roof to be removed later.

Of the relative merits of iron, wood, and masonry for underground work, much has been heard. Suffice it to say, that in Europe, where the utmost care is taken to preserve the timber,—by replanting for each tree cut down,—iron and masonry are put to extensive service. As to props of metal or of wood, the number would be the same—more for a brittle, less for a flexible roof; and whatever the condition of the roof, the size of the metal props would be nearly the same. Iron props are of the + or ○ cross-section, set on the thill or upon a foot-block, to be drawn by lever or bar and chain. Jack-screws have been used, but their expense is too great for their general use. Cast-

iron props, auxiliary to pack-walls, 9' apart, 5' long, 4" outside diameter, weighing 150 lbs., have been employed in collieries; their use is emphatically stated to be cheaper than wood.

Levels are not infrequently lined with iron tubing similar to that adopted for shafts. An illustration (Fig. 177) is given.

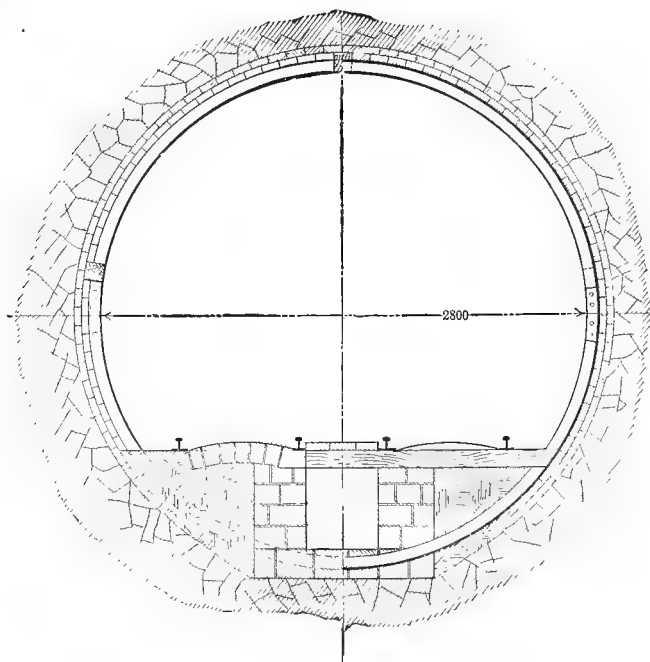


FIG. 177.

No system of timbering can be made sufficiently strong to prevent a general subsidence of the roof until the waste has been squeezed solid. Masonry for drifts and tunnels in this and the old country is very common. Where timber is rapidly destroyed, or where the pressure is too great for rectangular frames to be economically employed, the arch is pressed into service. On the other hand, good stone must be plenty and cheap. It is laid dry or cemented, with the walls straight

or curved and the top arched. Some mines dispense with timber altogether, and in others the masonry leisurely replaces the temporary framing. In the latter case, whether the masonry is

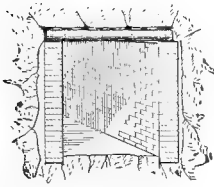


FIG. 178.

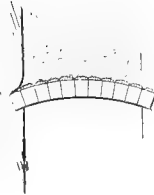


FIG. 179.

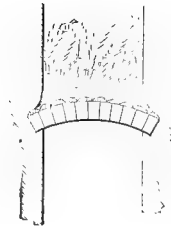


FIG. 180.

inside or outside, the timbers must be removed, and no spaces are to be left open, or to contain decomposable material.

The side walls are sunk into the floor 2' or 4' to a good bed, and with a batir if there is pressure from the sides. In New Almaden, Cal., mercury-mines 70,000 cu. ft. of masonry walling is built annually in the drifts. It costs four times as much



FIG. 181.

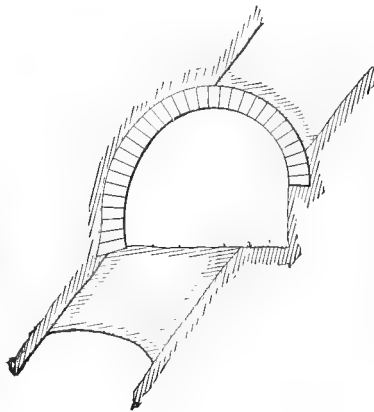


FIG. 182.

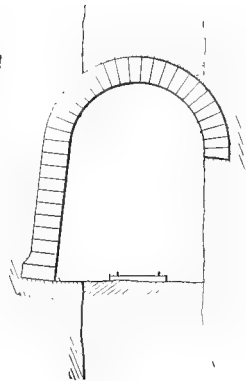


FIG. 183.

as timbering, but requires no repairs. The roof may be natural roof (Fig. 9), a timber cap (Fig. 178), or a pair of struts; but

the arch is preferable for a permanent way. The arch may be segmental, or, for heavy pressures from above, full centre.

Arches are used similarly with stulls, seldom over two brickstuck, and the centre in the axis of the lode. (See Figs. 179, 180, and 181). If one wall requires special support, a parting arch is frequently put in. When properly laid, arches will

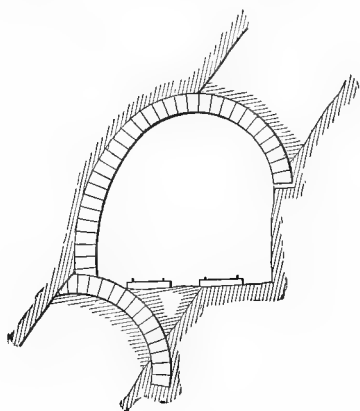


FIG. 184.

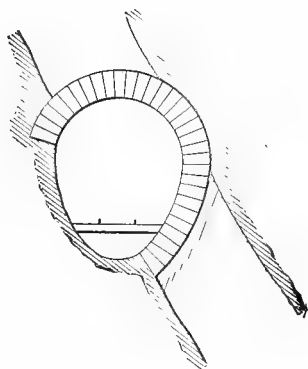


FIG. 185.

withstand the pressure which future emergencies may require. As it is not always necessary to use complete sets of timber, so only portions of the arch need be used. Where the vein or hanging wall is too weak for a cushion, Figs. 182 and 183 represent the arrangement. Imposts are always made for the

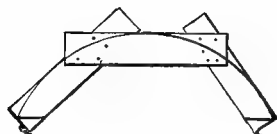


FIG. 186.

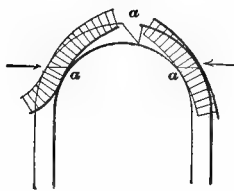


FIG. 187.

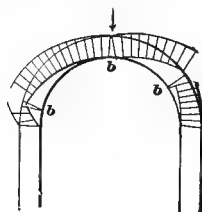


FIG. 188.

arch if firm seats cannot be had (Fig. 184). With a very bad hanging-wall some German mines use Fig. 185. The arches, of course, are built on centres, made easily of four pieces (Fig. 186).

The two accompanying figures, 187 and 188, may be interesting as suggesting the places of requisite strength for given conditions of pressure. If the pressure is from the top and any opportunity for bulging is given, the collapse will take the form of 188. If the side pressure is very great and the roof resistance small, the break is shown in Fig. 187.

Finally, in German mines will be seen the various examples of tubular walling of the elliptical (Figs. 189 and 190) or inverted oval form, the choice between them for greatest strength being still an unsettled matter. The latter gives greater width at the bottom and smaller area for pressure at the top. Masonry cannot be built unless the ground is previously timbered, or firm enough to stand while the mortar is drying. In soft ground the level is driven by spilling, which can only be replaced by the masonry retreating toward the shaft. When the temporary service of the timber has been accomplished, and masonry is to be substituted, the uprights are cut at the foot, the sills and spilling laths (Fig. 208) removed, the bottom arch is made first, side walls next replace the posts, the caps being

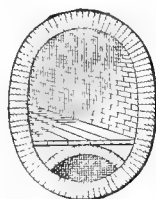


FIG. 187.

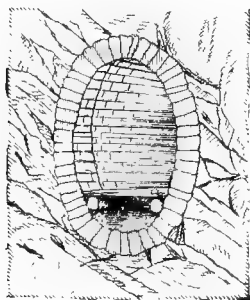


FIG. 190.

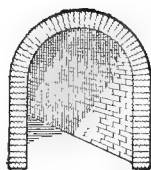


FIG. 191.

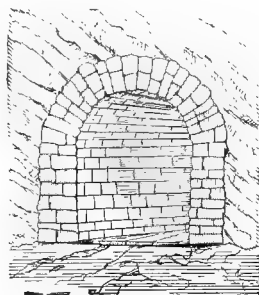


FIG. 192.

temporarily supported on props, the centre set up, and the top laid. Figs. 191 and 192 represent a masonry walling employed where the floor is sound.

Dams for keeping back water, formerly straight-backed (Fig. 193), are now arched, as illustrated (Fig. 194).

The use of iron is advocated and receiving ready acceptance in mining as well as tunnel work. The life of timber is short, its resistance low, and the component parts of the frame must be rigidly connected. The ordinary constructive forms of iron are applied in the ordinary way for columns, caps, or arches.

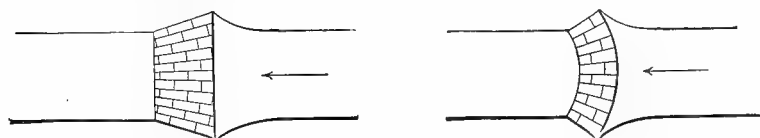


FIG. 194.

Their merits and methods are fully considered by R. Gottgetreu, in his "Baumateriellen."

68. In soft ground, which is liable to run, some form of stout framing is indispensable to prevent separation of the members. In the creviced matter of the Comstock mines, in the very poor ground of the Lake Superior region, in iron-mines where pillars cannot be trusted, in the rotten lead-ores of the Leadville beds, a most extensive yet simple system of framing has been introduced, which has found ready acceptance in various sections of the world. As a distinctive method of mining it was referred to in I, **12**, and though many properties are substituting a filling method, it still retains a hold on the mining public that qualifies it for a place here. Still, several causes hasten the decline of its popularity.

The plan is devised for soft ground, though it gives security in rock with a small tendency to cave, slide, or swell. The members are generally dressed square, and all to template for a faultless fit. For this work Hendey's or a similar machine (Fig. 195) is recommended. It is absolutely essential that the joints fit perfectly; then there is no swaying or buckling, and only an immense crushing can affect the frame. A slight play to each stick multiplies the opportunity for movement, and consequent destruction. The dimension of the members depends upon the quality of the ground. In the average soft material they are 6' long, giving a working face of that size, which having advanced an equal distance, is built up to. All

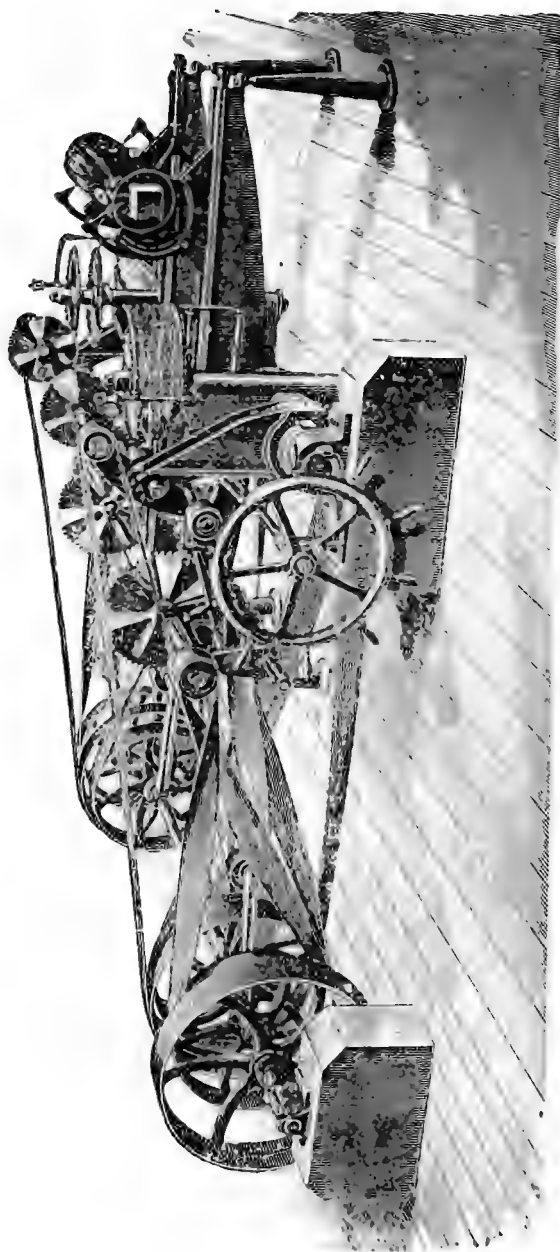


FIG. 104.

the sets are joined, and when properly cut the members act together to transmit the pressure throughout, and likewise to relieve one another.

Each set encloses a rectangular volume or cell (see Fig. 13), made up of sills, *s* (Fig. 196), cross-sills, posts, *p*, and caps.

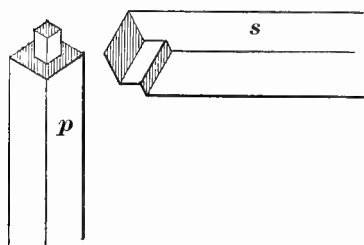


FIG. 196.

The posts are laid in the line of the greatest pressure, or else vertical, while the caps are at right angles to them. Flat mud-sills are laid on the floor, forming a square base, on the corners of which rest the posts carrying the caps. In the centre of each end of the posts

square tenons $\frac{1}{3}$ the area of the top separate the caps and sills, each of which rests on $\frac{2}{3}$ the area, and are of a length of $\frac{1}{2}$ the thickness of the caps. Thus a post supports two cross-sills and two caps, and rests on four sills (Fig. 13).

The frames are built up as fast as the work is opened, unless it happen that the ground will stand a while, until the timbermen can attend to the chamber. In ore that will not remain in place long enough to advance a set, a false set of cap and uprights, put in half-way, supports lagging overhead until the men reach the full length of a cut. If the ground will not allow of this advance, it runs, and only caving or filling is admissible.

The method of procedure varies in different regions, or perhaps with the nature of the ore. The lower floor is worked out first, the bents being added on at the right, left, and ahead; after which the next tier is set in the same manner directly over the first. Or, a species of overhand stoping is employed (Fig. 13), whereby the floor tier is progressed only a set or two before the next upper is advanced, to be followed later by another set above, and so on up. The order is a matter of indifference provided a perfect alignment is secured. Where this method is used in steep veins, very careful surveying is necessary to carry up the tiers of the lower level to a line with

those in the upper stope. Rooms in the Cambria iron-mine, Lake Superior, are timbered 250 feet high. Sound, plumb, square sets are still to be seen in the East Vulcan mine and elsewhere in the Menominee range, that were put in in 1881. Sticks 10" diameter or 14" square are used for 6' sets, and as large as 18" square are not uncommon, with 9' posts.

For platform, or under a scaling roof, temporary lagging is placed. Under the permanent roof or hanging-wall lagging is laid, and the space packed as close as possible with waste after the framing has been wedged tight.

Where the posts cannot be placed in the direction of the greatest pressure, diagonal braces are trimmed for, and driven into, the bents to take the inclined pressure.

Wedges are made of waste lumber in blocks 18" long, ripped into pieces 4" square. Then each piece is sawed diagonally from one edge to the other. Stulls cost in Leadville 6 cents per running foot; lagging, $1\frac{1}{2}$ cents per lineal foot; wedges, about 1 cent each; and head-blocks, 10 cents. The cost of a square set of 14" sticks is about \$4.50 entire; lining and placing, \$1.50 more.

In swelling ground it is customary to have some weak spots or frames that can be easily replaced. The swelling of felsite and trachyte will carry everything before it; roof is crushed, timbers are driven into the floor. Should movement commence, and any sill or range of sills squeeze or crack, the men should be called out at once, or the series attended to without delay. The engineer should "plumb" the framing occasionally in watching for any movement. The rupture of a single stick causes an increase of thrust upon the remainder, and ruin spreads rapidly. A chamber 90 feet high, with 13 tiers, was in ruins thirty minutes after the first warning.

Rooms and abandoned large stopes are often supported by a massive column of heavy timbers carried to the roof and filled with waste. These "cribs" are also built as an abutment in a chamber having the wall as a back. A good foundation is prepared, of the proper size and shape; two logs are laid parallel, and upon them, in the notches at the ends and at

the middle, three cross-sills are laid; upon them again rest a pair of sills slightly inside and above the others; upon these, in turn, another layer, etc., etc. Inside of this space, as fast as building, waste is piled. The logs are 10' and 14' long, and 12" to 20" diameter at the butt. Cribbing may also be built with the removal of the ore, stoping overhand, and utilizing the waste to fill the crib (Fig. 197). In abutment cribs the cross-sills rest on the waste at one end, and are held in place by the weight above them. It is questionable if there is any

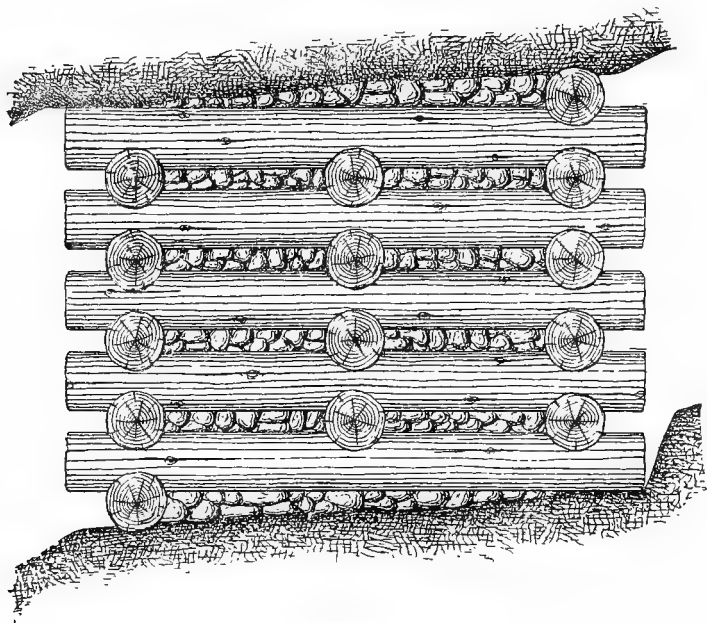


FIG. 197.

choice between the square set and cribbing in large rooms. The former is well suited to turning off into small cells at the ends of the room, but it is dangerous if side-pressure exists. Cribbing will never do in very soft ore under brittle roof. This arrangement constitutes a very strong "made" dump, where the rock is not permitted to roll away freely,—as for instance, on account of contiguous buildings down the hill.

The securing of underground chambers intended for steam-pumps, hoisters, stable, etc., is not an easy matter. They may be built in any suitable shape that provides sufficient room. Being large, great skill is required to utilize framing or walling materials to the best advantage. Undoubtedly an arched roof will give the greatest resistance and strength, and masonry is therefore suggested. Besides, the hot, damp atmosphere of steam would rapidly destroy timber. Still, the latter is more convenient than masonry, in heavy sets, butted against each other, and spliced, with a key-block introduced. They are laid close together, lagged over, and packed to prevent wedging apart. The ventilation of these rooms should receive special attention. For air-compressors, engines, and coal-cutters an enlarged level will do, with some stouter caps on the wall-posts. Railroad rails and I-beams, log-lagged, are also used.

The styles for timbering plats and landings have already been suggested. The main difference at these points from that throughout the level is due to the intersection of two prisms or cylinders.

The styles of the timbering must vary with the character of the intersections. The frames must support one another as well as the country rock, and give firm fastenings for the plats and doors used for the landing of the cars and buckets. The level or gallery should be widened near the shaft, to give room for sidings, storage closet for powder, and steel. The sets of the shaft timbering opposite the level opening will be lacking one plate each, or two, if the level crosses the shaft. In this event, two very heavy stulls should be hitched at the bottom of the level, and two at the top of gallery, between which are heavy corner posts carrying the end-plates and the rest of the sets.

The masonry lining of a shaft is supported by an arch to, or by lintel on, two posts or walls at the sides of the gallery. The former gives a high opening for landing the buckets.

Mill-holes carried up with the waste are solid cribs of 30" square for a man-hole, and may also have a compartment for sliding ore (Fig. 4). Either cordwood or sawed blocks are

used for the lining. The latter plan gives them greater durability, for the abrasion is along their cut faces.

The timbering of slopes is similar to that of galleries, except that greater care is taken in cutting.

The timbermen's tools are few. Generally speaking, they need be only hatchet, hammer, and wedge, with a bar and chain for casual work. The timber should be delivered below ready for insertion. A sawmill at the surface is now as much a component of the surface improvements as is the boiler. The Hendey machine is an excellent framer for round or square logs (Fig. 195). The saws being adjustable in every direction, it may be set to cut the required size. The ends are cut simultaneously. With three movements by the laborer and three positions of the log the tenons and shoulders are cut accurately, and at a cost of but 18 cents per set, as against \$1.70 by hand-work.

Though the work of timbermen cannot accurately be stated, the following is given as the experience of an "old miner" for one day's labor:

Making about 100 feet of wooden ladders;

Placing 65 sq. ft. of close shaft-lining down to a depth of 70 ft. ;

“ 40 “ “ “ “ “ “ “ “ 200 “

Cutting and dressing mine frames, 50 running feet;

Framing mine-sets 40 “ “

Dressing sills, 30 “ “

Making 60 poling planks;

“ 100 “ wedges;

Setting up 20 pairs of props and head-blocks.

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CHAPTER IV.

DRIFTS, TUNNELS, AND ADITS.

69. Utility, dimensions, and location ; mode of driving, progress, and cost. 70. Tunnelling through hard and soft ground ; dimensions for various purposes ; difficulties in soft rock ; description, and comparison of the English, Belgian, German, and Austrian methods ; the American method ; examples of long tunnels ; auxiliary shafts. 71. In treacherous ground ; method of spilling by laths ; by wedges ; poling ; Durieux's method ; iron shield and pneumatic processes ; masonry for permanent security ; principles in the construction of arches and centres. References.

69. DRIFTS, levels, galleries, or gangways, according as they are driven for temporary or permanent ends, receive a corresponding amount of care. The manner of driving does not differ much from that of shafts, except that, as their area is less, and the water encountered can flow away freely, the difficulties are not so great as for a permanent hoistway which traverses a variety of strata. The locations of these drivages are determined by the considerations affecting the method of mining (see I, 6).

Before locating a tunnel of any importance a careful study of the ground is requisite. All the data obtainable from geological reports, borings, etc., should be availed of. The character of the strata, their pitch, and the direction of the subterranean drainage should be known. They are of decided service, though not absolutely decisive. When the strata are recognized, the kind and amount of timber required may be ascertained.

Cross-cuts, connecting the shaft with the vein, are rarely timbered, for the country rock will generally stand during the period of their utility. Their size is the same as that of

the drifts, or, for a busy level, wider, for siding and storage. If they require protection at all it must be by masonry or iron. Adits and levels are in the lode, and secured as shown in the previous chapter. Usually the stull suffices for the average length of time the drift is kept open. In underhand stoping only the trackway requires support, below which nothing but reachers, 5 feet apart, are needed for the work. Gangways and galleries have a larger area, and, being exposed to more treacherous conditions, possess better examples of the timbermen's art. The securing of them by masonry or iron is optional with the owners. Tunnels for railway or drainage purposes must be walled. In all cases the drifts should be driven to a normal profile at once without requiring subsequent trimming; the corners of intersecting levels should be rounded off, to ease haulage and aid ventilation.

Uncreviced hard rock may be penetrated without serious drawback. Granite, dolomite, and gneiss are hard drilling, but

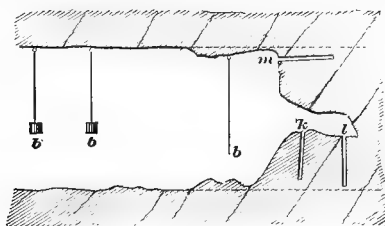


FIG. 198.

offer good roof, and are dry. Slates and shales are bad, and require arching or block timbering. Porphyry is treacherous, because, though hard when first opened into, it soon decrepitates on exposure. Some of the limestones and sandstones are porous and wet. Clay seams are bad primarily, besides being watercourses for the upper porous strata. The most favorable material is that which is sound and durable without being very hard, though hardness presents no special difficulty beyond an increased time and cost.

The alignment is maintained (Fig. 198) by three plumb-bobs hung from the roof. They pursue an almost dead level,

though 2 feet per 100 is an accepted grade in metal mines. In collieries, the question of grade is involved with the proposed system of mining. The attack of the heading is over the entire face, unless the size of the way or the softness of the material may require its attack in sections.

In uncreviced rock, the order of the breaking is immaterial, so long as a good bench is had to shoot from and a good "leave" is obtained for the next shot. Even this is a matter of indifference, since the introduction of machine-drills and simultaneous firing, with a system of drilling, somewhat as explained in 90. A drift of ordinary size will easily accommodate one drill, while two can advantageously work in a heading 10 feet wide, obtain more angling holes, and advance more rapidly. In railroad tunnels four are simultaneously drilling. Hand-work is much more

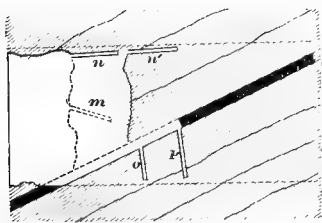


FIG. 199.

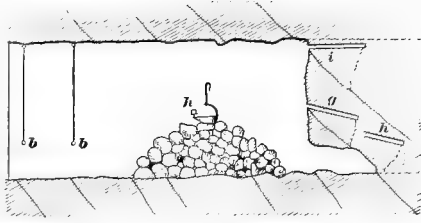


FIG. 200

depended upon for the driving of gangways, but machine-drills are becoming popular. Undoubtedly there is gain in time; the progress by machine is greater than by hand, even if the actual cost be the same per lineal foot—which it is not always (see 87). Where the drills and air-compressors can be put to use after the mine is opened, it certainly would be advantageous to employ them in development.

Drifting into pitching and stratified ground is simplified by the assistance afforded to shooting. If the seams pitch toward the face (Fig. 199), the upper holes are fired first; or, in high galleries, the central holes will make a good bench to shoot to from top to bottom. If the cleavage is not marked, or away from the face (Fig. 200), the bottom holes are fired before the

upper ones. With electric firing, the sequence of shots is of little importance; besides, as the driving is usually done by contract, the mode of working varies with the individual. Three men form a gang in driving, and can manage the ordinary-sized drift. In the gangway of 7×7 two machines or two pairs of cutters have ample room. Their progress cannot generally be stated. In hard rock a foot a shift is fair, while soft can be penetrated at the rate of three or more feet a day. In shaly ground greater progress might be made did it not require good timbering. In soft ground the advance depends upon the skill of the timberman. The consumption of steel in medium ground is about 25 cents per cubic yard removed, though pink quartz will dull 150 bits to the hole, and the blacksmith will consume more steel than the rock. The consumption of powder is about \$1 per cubic yard of medium-tough rock removed. This amount will vary with the area of the face, and whether the breaking is done by simultaneous or single shots.

70. Levels with a greater height than 8 feet must be broken in benches. Railroad tunnels are of this description, and in hard rock may be driven without any temporary timbering, the benches being attacked like stopes (Fig. 201), where the numbers express the sequence of openings. Frequently, in driving long tunnels, the drift No. 1 is pushed as fast as possible in order to make connection with a shaft or a similar drift approaching from the opposite direction. This communication is for ventilation and haulage purposes.

Numerous tunnels have been driven more than five miles for mining purposes. The Freiberg is 24 miles long; at Clausthal is one nearly 11; the Joseph II. is $9\frac{1}{2}$; the Ernest August, $6\frac{1}{2}$ miles; and the Sutro, 5. These have, moreover, lateral branches that enable them to subserve a great territory. The Gwennap adit, in Cornwall, is said, with its branches, to attain a length of 30 miles. The greatest depth below the surface is not over 900 feet for any one of these.

Progress is facilitated and ventilation obtained by the sinking of shafts at convenient points along the line to the tunnel

level; and from them the excavations are begun contemporaneously with those at the mouths of the tunnel. Sometimes the shafts are to one side of the tunnel line, to keep them free and clear of the tunnel-work. The amount of time allowed for the completion of the tunnel determine the number of points of attack and the number of shafts to be sunk. The latter is also dependent upon the cost of sinking and the hoisting through them relative to that of the long haul in the tunnel. These relations can be mathematically expressed when the several aspects of the case are given. From Foster's Callon's "Lectures on Mining" the following formula is taken:

$$Q = xq + (PS + P'S')\frac{l^2}{4r};$$

wherein q = the total known cost of shaft, x = number of shafts to be opened over a length l , S = the area excavated, S' = the section of the lining, D = the distance between the two adjoining shafts, P and P' = the cost of haulage per lineal yard for each cubic yard of rubbish and of walling material.

$$D^2(PS + P'S') = 4q.$$

The St. Gothard Tunnel, 48,840 feet long, was run from the two ends only; so the Mont Cenis Tunnel, 39,840; the Hoosac and the Sutro tunnels with the assistance of two shafts. The Washington Tunnel, 20,715 feet, had four working shafts. The Rothschenberger mining adit had 18 points of attack along its 24 miles of length. Rziha estimates that the "additional cost of running headings from a shaft is from 5 to 10 per cent higher than running from portals." He also gives a unique table to show that the rate of progress per month is apt to be greater in long than in short tunnels. Of those less than 100 m. long, 29 feet advance was the average of 3; 13 between 400 and 600 m. showed average of 57; 8, up to 1000 m., 114 feet; while 6 between 3000 and 4000 m. had an average of 219 feet, and 4 over 4000 m. progressed 259 feet per month.

Provision for traffic is not a simple matter when it is recalled that, besides the actual mining of an area (say, for example, that of the St. Gothard Tunnel), 26×20 , at a rate of 18 feet per day, and the placing of 1000 cubic feet of timber and 300 tons of masonry per day, 750 tons of broken rock, 15 tons of lumber, and 300 tons of masonry must be handled, loaded, transported, and unloaded at the same time.

As the hoisting through shafts is about half as fast as haulage through a heading, the use of auxiliary shafts is not so economical as it was before the present development of machine drilling and blasting methods.

There are other methods of attacking the tunnel face besides the one depicted in Fig. 201. The two side drifts at the

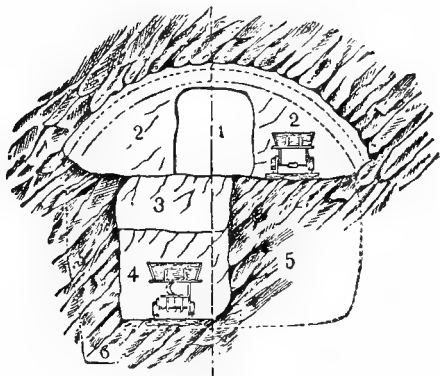


FIG. 201.

bottom may precede the main excavation; in these are built the walls for the roof arch which ultimately lines the tunnel. While the small drift, 1, is being driven to the connection, the sides 2, 2 are following, and in firm ground they may precede 5, 5 at the bottom by 150 feet. At this point the tunnel is being lined with timber or masonry while gangs are also breaking ground on benches 3 and 4.

If the roof needs support, the plan of work contemplates one similar to that of Fig. 202, with the exception that the central core, 3, 4, remains as a support to the timbers arranged

as shown below. Inside of this frame the masonry may be completed, after which the inverted arch, if required.

In soft ground, and especially in inclined strata, the order of driving the benches and the amount and character of timbering varies according to the dip. Circumstances and difficulties are so diversified that no uniform infallible rule has been established for the guidance of the engineer—probably because no system has a superiority over all others for any and all

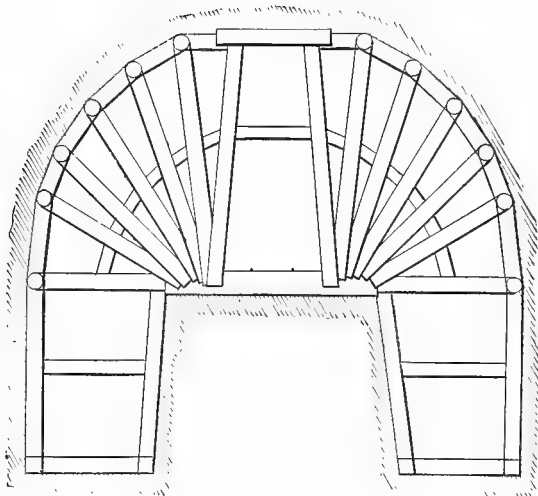


FIG. 202.

conditions. Safety is an element as important as time and money, and often is a determinant.

We have four systems of tunnelling, known as the English, Belgium, German, and Austrian. A brief review of them is taken from Drinker's "Tunnelling."

The English, developed from the Thames Tunnel difficulties, consists in taking out the full area at once, after the preliminary top-heading has been made, and in supporting the roof by longitudinal top bars while removing the lower section. This gives a full clear area for putting in the masonry. After which, in material having any tendency to swell, the space behind the side walls is securely grouted. Though built

quicker and applicable in 90 per cent of cases, it is unsatisfactory in very bad ground; in heavy ground it requires more timber than does the Austrian, its strongest competitor.

The Belgian method, introduced after the iron shield had been tried ineffectually in quicksand, builds the tunnel as an open cut down to the springing-line of the arch. The arch is then laid, recovered, and underpinned, the bottom removed in benches, working downwards, and the abutments built. Sometimes the centre-core is left, though that is only done in the French and German modifications. The entire area is not attacked at once, but divided into several benches, each being worked separately. The underpinning of the arch may be safe enough in hard ground, but it certainly affords a doubtful security in loose material.

The German system gave rise to what may be called the centre-core system. The work of excavation begins at the foot of each abutment of the arch, where a small heading (Fig. 202), with timber sets is first driven. In this the foundation is laid; above it a second heading, large enough to build another height of wall; above this another, in which the masonry is carried up. The top is then excavated across, and a connection effected between the two sides. In this the arch is completed without the use of centres, while the roof is being supported by stulls or props. The core is then removed.

In hard ground it gives cheap working, from the fact that the core has several faces of attack. In soft ground it is safer, because of the small exposure of roof and face; and the centre-core saves timber. Its ventilation is bad, and the cost of laying masonry is larger than where the masons have elbow-room; it is hard to securely timber, and several prominent engineers have decided against it. Certainly, in soft, treacherous ground, like shales and clays, its defects disqualify it.

The Austrian distributes the mining over the whole area in small sections. First a bottom heading is driven and afterwards connected to a top heading which is finally widened to full width for a bar-timbering which is carried down to the springing-line. Sides are excavated for the walls, after which

foundations are prepared and the side walls built; finally, the invert arch. The cross-rafter timbering (Figs. 203 and 253) used for support admits of the transfer of pressure, and there is no such undue concentration, as in Fig. 202. This disposition of the timber affords a greater strength, and is the leading feature. Additional braces are sometimes added, and the

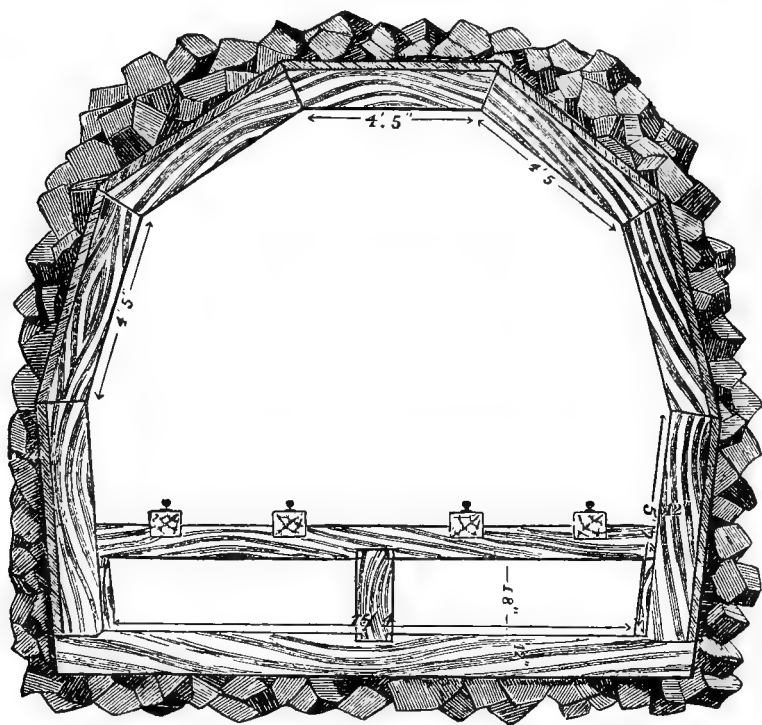


FIG. 203.

sets connected every which way; but the design is to arrange the timber of each section in such manner as to form an integral part of the completed system. After the roof-timbers are in place, plenty of space is left below for masons, and good ventilation is had.

Each of the four systems named have distinctive features commending it to favor under given conditions, and a brief comparison may be given.

The centre core system is emphatically inapplicable for loose rock where other methods are better. It is quite expensive in all of its operations, and is not safe.

The English system is safe enough for most tunnels that are likely to be driven, and offers ample space for the miners and masons to work, but it is not adapted to very bad ground or where heavy pressure is encountered.

The Belgian system seems to be preferred for moderately firm ground, in which it is quicker than either of the two above mentioned; but it is hardly suited to running ground because of the great care which must be taken with the underpinning. The cost of transportation while building is higher than in the others because of the frequent interruptions caused by the transportation scaffolding. In very loose ground it is positively dangerous.

The Austrian system has more to commend it and less to condemn it than the others. All the driving operations are cheaply and easily conducted. The driving is least interrupted, and safety is assured if the middle timbering and the longitudinal bracing be well apportioned. The forepoling, if employed, would be in the direction of the tunnel axis, and the masonry arching is not delayed.

In Figs. 205 and 206 are illustrated the mode of "block timbering" permanently placed in the tunnel. Each of the nine blocks is 10" $\times$ 10", the legs being 12 inches square. From the three-rafter system of roofing the tunnel American engineers developed the block-timber arching for the support of the roof rock which is very loose and requires close bracing. The wooden voussoirs were first used for temporary support only, until they were replaced by brick or stone arching, but latterly they have served as a permanent arching lined with a fire-proof sheeting for railroad use.

We cannot claim any system of tunnelling as our own, for neither the number of tunnels nor the difficulties encountered

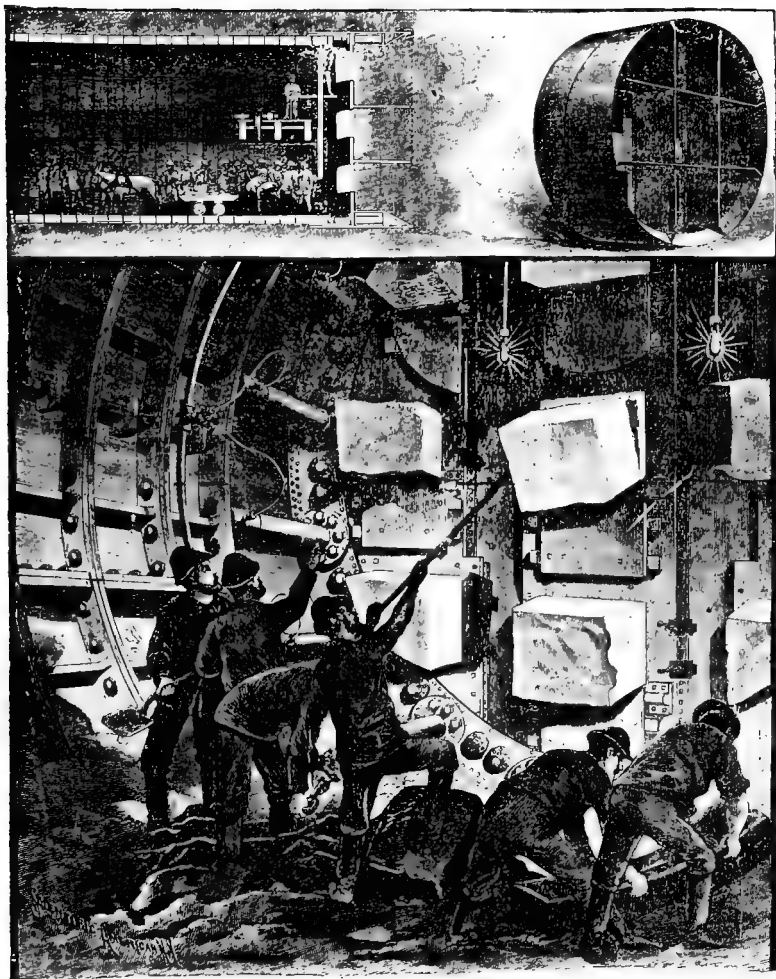


FIG. 204.

are as great as in the Old World. The Austrian method is the nearest approach to ours, or, rather, is the one which our engineers have adopted with a modified framing. The mode of driving and timbering is illustrated in Fig. 201. The upper heading, 1, with its enlargement, 2, 2, precedes the work upon the "bench." With a tunnel area of $21' \times 27'$ the heading is about 8 feet high, and the bench, the remainder, attacked in one section. The upper heading is timbered, rafter style, three or nine voussoir bents forming a block-arch on heavy permanent pole standards, or on the side rock if sufficiently sound. In loose rock the number of bents increase and the timbering is heavier. Inside of this frame arch the masonry is built. A segmental arch 21 feet high by 18 wide in the clear had nine voussoir-blocks 25 inches long and 10×10 inches section; two wall-plates, 6 feet 4 inches long, 12 inches square; two posts, 14 feet 1 inch $\times 10 \times 12$ inches; and 45 lagging, 6 feet 6 inches $\times 6$ inches.

Fig. 203 illustrates the style of bar-timbering, which, however, will not permanently withstand much lateral pressure.

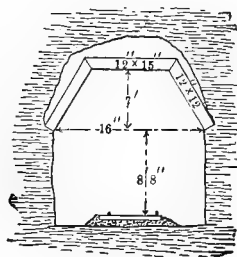


FIG. 205.

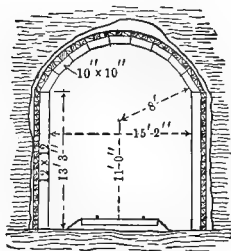


FIG. 206.

71. In running ground or decomposed rock the area attacked is limited by the rapidity with which permanent support can follow the excavation, during which provision must also be made against the pressure coming from all sides. In no region, not even in the Rocky Mountains, is the engineer free from the liability of striking ground that may overwhelm the miner before timbers can be inserted; so that the ground has

to be restrained up to the face as well as behind the frames, with which close lagging may suffice to prevent movement. If it is not in new ground, the consistency of which will determine the details of its penetration, it may be where timbers have rotted or given way that a cave or a run may occur.

In alluvial or sand the method of spilling or a pneumatic system is indispensable. Varieties of the latter were employed under the waters of the Thames, 11,880 feet long; Severn, 22,992; and the Mersey, 23,760 feet, where the inrush of water amounted to 20,000 gallons per minute.

Spilling by laths is a method applicable to shafts or drifts equally. In front of one stick of a set, and behind its mate in the next advancing set, pointed heavy planks are driven, one set in advance of the face, close together on one or all of the sides from which the pressure is exerted. The fore end of the plank is forced down upon its set, while the rear end is held against the lower side of its cap, being protected from pressure by the previously driven upper lath. This is the method of forepoling illustrated in Fig. 207. The progress in soft, heavy timbered rock is three times as fast as in very solid rock.

The "spilling" protects the fore-breast by horizontal laths, as long as the breast is wide, held against it and braced to the nearest set. Each lath, *a* (Fig. 208), is removed in descending order to permit some ground to run off; it is advanced a short distance and braced again, *b*. The progress depends upon the speed with which the spaces may be opened and closed; as these are small, the movement is controllable, and there need be no fear of sudden shock to the timbers. The method is simple, and has been eminently successful under many circum-

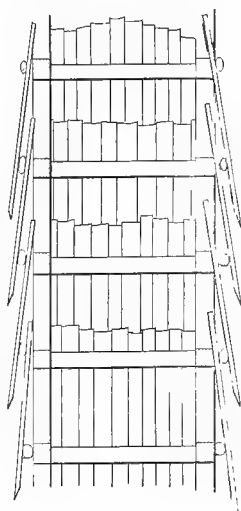


FIG. 207.

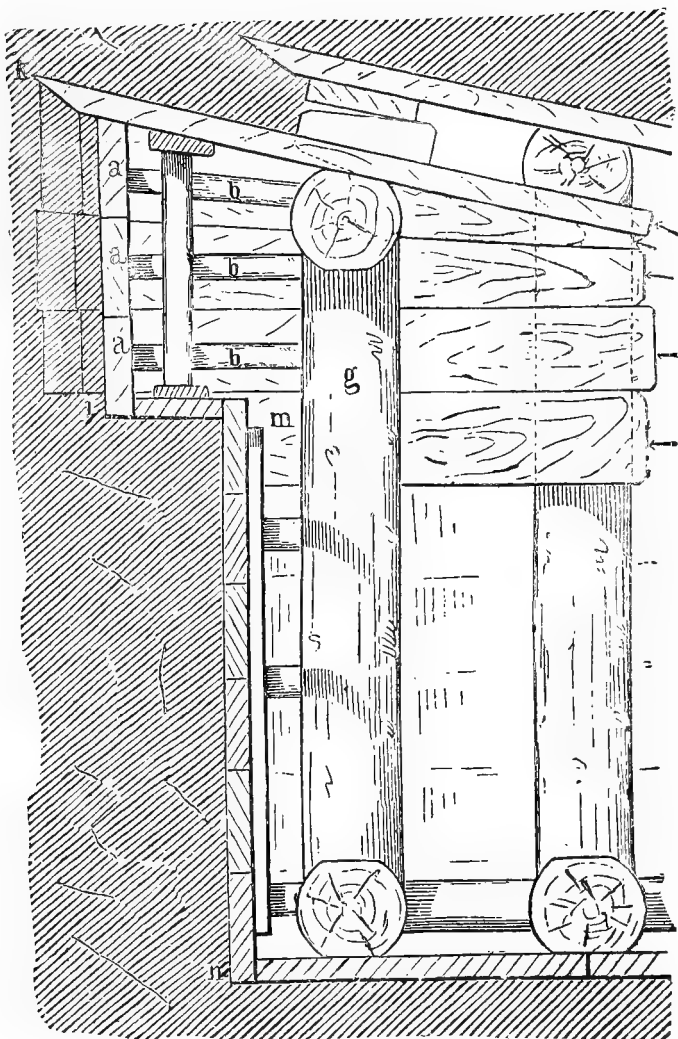


FIG. 208.

stances; once, with masonry lining following the forepole, a tunnel was executed within 18 feet of a river bed.

An entirely different principle is that employed by Durieux and others in Westphalia, whereby the ground was forced ahead of the work, and was not removed at all (Fig. 209). The

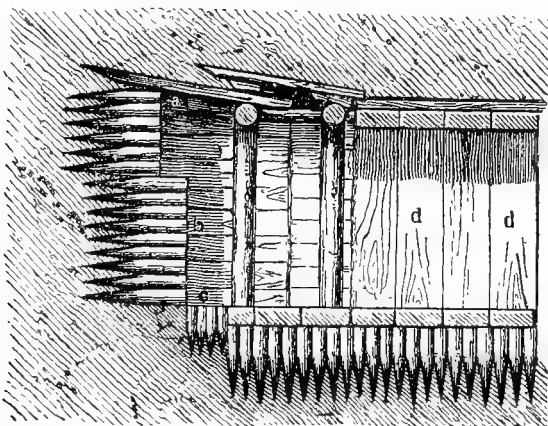


FIG. 209.

walls and roof were forepoled, but the breast and floor were checkered by pyramidal pickets, completely covering the exposure. Those on the face were square faced, and larger than the floor-wedges, which were 12 inches long and 4 inches diameter, driven by mallets. The floor-pickets remain in permanently, while those at the face force the soft material ahead and advance in this manner. The battering they receive renders them useless in a very short while, when they are replaced. There is no material to be hauled except that used in the construction, and rather bad ground has been thus traversed. On occasion a short lateral drift is similarly pushed to relieve the main work. Four men in a drift 5 feet $\times$ 6 feet in morainal matter will advance 4 feet a shift.

The finest piece of tunnelling is the construction of the new Croton Aqueduct, where water, mud, quicksand, and all varie-

ties of loose soil were penetrated in an area of 676 square feet. Forepoling for the roof and sides, picket-spilling for the floor, and the American system with block-arching were adopted at various points. The ground in extended places was so bad that 24-inch timbers were crushed by the pressure.

Iron naturally suggests itself as a safer constructive material for loose soil; and in 1825 Brunel put it to use for excavating an area 38×22 under the Thames River. As in the spilling method, the face was covered by laths about $3 \times 3 \times 6$ inches, and for rapidity of work was subdivided into three tiers of 11 breasts each (Fig. 210), each being protected by a cast-iron shield, from which struts hold the laths *d* in place. One man to a cell operates by taking away a lath and replacing it 3 inches in advance. The laths are removed successively downwards until 3 inches of progress has been made, after which the alternate frames are carried ahead 6 inches and the performance repeated. In each cell is a similar performance. The masons follow the miners very closely at *G*.

The second Thames Tunnel was completed by the use of a shield, which with pointed shoes was forced into a stiff clay at the rate of 9 feet per day by jack-screws exerting a pressure of 60 tons on it. Three men crawled through a door in its face, and excavated some earth preparatory to the next move.

The practice at the present time, by which long subaqueous tunnels are executed, is a variety of the pneumatic system. A pilot-tube or caisson penetrates the soil, which is held back by compressed air. The masonry, perforce, is built as fast as the shield, tube, or caisson advances. In the Greathead system a cylindrical iron shield, 21 feet in diameter, is thrust from the masonry by hydraulic pumps under a pressure of 3000 lbs. per square inch. The front end of the shield has a heavy ring-shoe, while the rear end encloses the masonry. The silt squashes through the doors opened in the face of the shield. Only about one half of the silt is trammed out; the remainder, mixed with one fifth water, is taken out by aspirator.

The Anderson system has a pilot-tube only 6' diameter, with timbers inside resembling the spokes of a wheel, and pre-

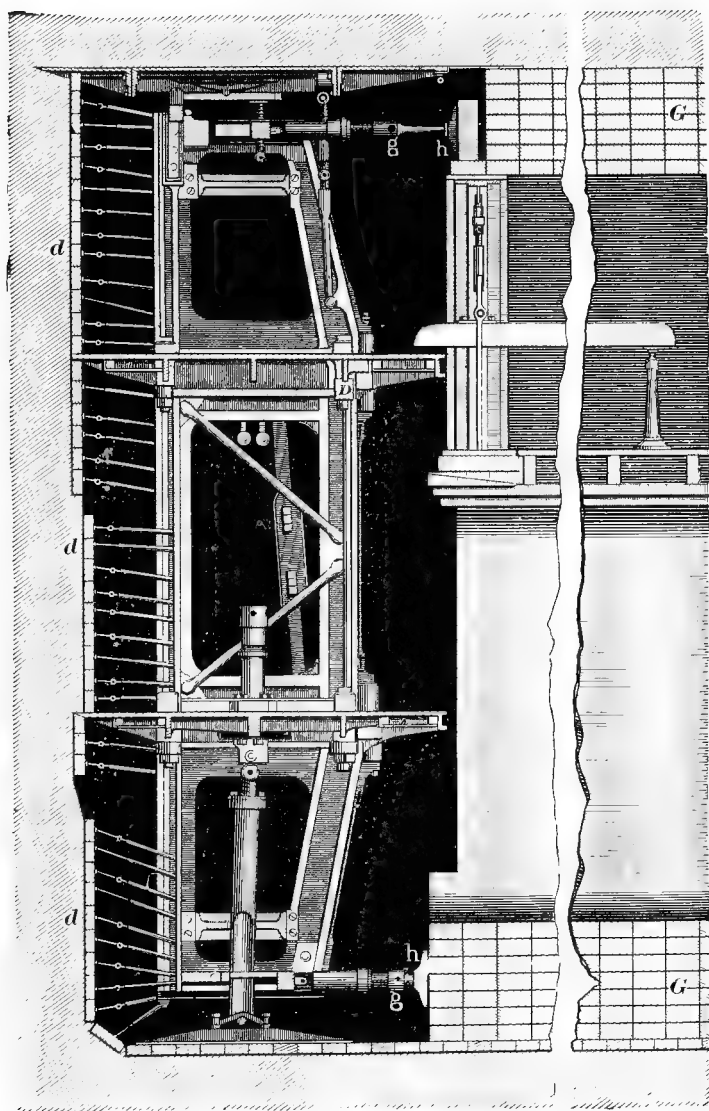


FIG. 210.

ceding the main work. It is of $\frac{1}{4}$ " plates, 12" $\times$ 24", riveted together by means of flanges; and when a cut has been excavated into the heading large enough, one of the plates is placed and held by props (often the plates are held by compressed air during the work), on each side other cuts are made for two more plates, which are riveted to it. Rings of the pilot are thus successively completed.

Around this, in small terraces, and considerably behind the pilot, the main shell, 17' in diameter, is finishing in a similar manner, the plates being propped from the pilot-tube, which is always braced from the masonry that lines the shell. With its progress the rear rings of the pilot tube are removed and their plates shifted to the front end. The masonry consists of six courses of brick laid in cement. To reduce the volume of the tunnel that is kept under the compressed air, brick bulkheads, 4' thick, provided with two air-locks, are built every 400 or 500 feet. Only the two nearest the work are maintained.

Whatever the procedure, the masonry is built on centres and by template, for invert and walls. The centres should be made of light, small, easily-framed sticks, that are not so close as to interfere with work, yet strong enough to support the thrust that may fall on them when the tunnel-timbering is removed. Its shape may be whatever is the most convenient for the traffic. The elliptical linear arch is, however, the form most commonly adopted, the side and roof comprising the upper part of the ellipse, which is closed below by a segmental invert arch, with the springing lines on horizontal faces. In stratified rocks, the strongest form for the roof is that of a pointed arch. Sometimes in solid rock the horse shoe form is used for the top and sides, the floor being level.

In preparing to tunnel silt, both the weight and the vertical pressure of the overlying material and the lateral movement of the loose paste are to be resisted. The first is a matter of determination, and the ability of the completed structure to withstand this is also a matter of mathematical calculation; but the second is the difficulty to be apprehended. The

Hudson River Tunnel engineers, however, satisfied themselves that the tendency of the gravel to pour over the top of the tube would reduce the lateral stress to the resistance of the tube. In Fig. 204 is illustrated the hydraulic shield which was established as a very valuable means of penetrating soft materials by the results of the St. Clair tunnel. Certainly the experience of the engineers upon subaqueous driving goes to show that it serves better than the timbering systems as regards the prevention of overhead settlement. The liability to settlement in front of the shield may be overcome by grouting under pressure at the rear of the shield between the lining and the roof.

The operations of sinking and the timbering of a slope are similar to gangway-driving, except that the sill is indispensable to the set. It must be well bedded, and let into the rocks on each side to prevent the roadway from slipping down hill. Ofttimes it is stayed by plugs driven into the floor. The sets are also braced against one another by longitudinal studs, between the posts, at their head.

Before locating a tunnel the ground should be well studied from geological reports, borings, and such other information as may be procured. Maps are serviceable in showing the important features; and a systematic plotting of all data, geological and otherwise, gives a good basis for conclusions. In Drinker's "Tunnelling" will be found a discussion of the geological conditions affecting tunnel locations.

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CHAPTER V.

BORING.

72. Punch-drills for artesian and oil wells ; history of its advancement ; accounts of deep bore-holes ; Fabier, Kind, and Degousee tools ; Mather and Platt system ; description of an oil-well plant. 73. Spudding, cost, progress, accidents, etc. ; tools, rods, torpedoes, tubing, and their recovery ; where used in preference to the diamond-drill ; novel Colorado method.

72. BORE-HOLES for testing the nature of the strata below may be drilled by one of several methods. They may be driven to prospect for gas, to afford an outlet for water (p. 145), for pumping brine (p. 28), or to subserve many of the precautionary measures of mining, and they may subsequently be utilized for ventilation or the tail haulage-rope. Many holes of over 2000 feet have been bored for commercial or scientific purposes, the largest being that of Sperenberg, 4151' of 14" diameter. In these was observed the increase of temperature with depth (p. 240).

The evolution of the punch-drill, as it is called, may be traced through several stages, as the difficulties of weight, vibration, breaking of the tools, and keeping out water were one by one overcome. The primary idea of a cam operating an oscillating beam, suspending the tools, has not been much altered. In primitive days the tool was manipulated by a spring-pole and hand labor, or by a foot stirrup,—giving rise to the expression of "kicking down a hole,"—and served for moderate depths.

The drills were originally hung by rods ; but their use gave rise to various troubles that were not easily remedied. Every 600 feet of 1" rod weighs a ton ; and the rapidly accumulating weight for deep holes became serious. The jars produced

concussion that injured the material, and loosened the joints; and the repeated breakages of rigid rods proved fatal. If iron rods are used, they should be long, so as to reduce the time occupied in screwing and unscrewing; or else the derrick must be tall (Fig. 185). In cross-section the rods should be square. All other forms have failed to give satisfaction. It is the simplest and the most easily handled; 1" square will do for ordinary depths.

Both the screw and the sword-blade joints are used. Care should be taken that the socket is on the lower end of each joint, or the sand will give trouble by jamming into it. The introduction of the hollow rods instead of solid ones gave great progress with the same power, and tapered wood rods, substituted for iron, were an advantage; but rope is the lightest connection between engine and drill with which our artesian and oil wells were dug. The Chinese were ages ago ahead of us in its use.

The Mather and Platt system of connecting the borer with the motor by a flat hempen rope is still used in Europe for deep holes. An ingenious device was added to rotate the tool, since this could not be accomplished by the twist of the rope. A movable collar, cut with inclined teeth at both ends, played two or three inches vertically in an iron bow, which had above the collar and below it sets of corresponding teeth, one half tooth apart. With each drop the collar engages the teeth of the lower set, and turns it and the chisel below one half a tooth; rising, the collar is turned one half a tooth by the upper set.

The evils consequent upon the blow were remedied by appliances similar to the "jars" now used (Fig. 213). Oennhausen's chisel had at the top a four-armed projection which played in corresponding slots of a cylinder, which terminated the rope. The rope was lowered and turned slightly; the ledges of the cylinder thus slipped under the cross-arms of the chisel, which was caught by raising the rope and its cylinder; a side-jerk freed the tool, whose cross-arms slid in the slots, and dropped it a distance equal to the length of the slots.

A pair of bent levers acting like pincers, grasping the tool

and raising it to a proper height for a free fall, is the form of Kind's invention.

The tool to the development of which the oil and gas discoveries gave impetus is called the jar. The chisel, with its auger-stem (Fig. 213), is connected to a yoke which can slide inside of a large, flat chain-link. These fall freely to execute the blow; but on the rising of the rope and link the yoke is jarred, and the chisel loosened and raised for the next stroke. The chisel is a straight or an X bit of the best iron, and tipped with steel. An auger-borer is used in clay; a V cutting-edge in hard rock. The set of drill-tools is made as long as 50 or 70 feet, to more readily keep the hole vertical.

A temper-screw regulates the feed to the rope and tool as the hole deepens. It clamps the rope to the walking-beam of the engine, and allows of a play of 4 feet. The débris is cleaned out; and samples of the rocks traversed are secured by a sludger, which is a plain cylindrical iron tank with a stem-valve at the bottom. (See Figs. 75 and 133.) To save time in changing, it is usually operated by a separate windlass from that winding the tools. This is also called a sand-pump, from the fact that the tank must be run up and down several times in the hole to fill it with sludge. The débris sampled should be vialled and labelled or poured into a glass tube for reference.

73. The operations of drilling imitate the jumper. By the Mather and Platt system the rope with its pendent tools were raised by a single-acting piston operating on a pulley, which on the up stroke raised the tool, that was allowed to fall on the descent of the piston. This is the feature of the plan, which is distinctive; its progress compared favorably with the other methods applicable to 14" holes. Only three men are employed. By removing the inside cutters a solid core could be obtained.

The spring-pole is operated by hand- or foot-power, though its rebound lifts the tool after each stroke. Holes over 3" diameter or 300 feet deep cannot be drilled with this crude machinery. Usually, spliced or strapped rods connect the chisel with the pole. To start the hole with this or the port-

able derrick-machine, a shaft is sunk through the surface-soil to bed-rock, and a 10" plank-box leader, held vertically, guides the tool.

The more complete outfit, with which 6" holes may be carried a thousand feet or more, consists of a tall derrick (Fig. 212) having a sheave at its crown and a 10-horse-power engine winding the rope on a "bull-wheel" drum, and operating the tools by walking-beam through a pitman. The height of the derrick should be sufficient to suspend the full "string" of tools. The rope is of hawser-laid cable about 1½-inch diameter.

Under certain conditions, "spudding" must be resorted to until the rocking-beam connection is made. While the bull-wheel is revolving, the rope holding the tools is tightened a little by hand and wound up somewhat; it is then suddenly relaxed, and the tool falls. Another pull again winds the loose coils, which drop the tool when slackened; and so on for 50 or 60 feet of hole. After each blow of 2' or so, the chisel is turned by a lever in the temper-screw. When the screw has been paid out, the tools are hoisted and the chisel sharpened, if necessary, or replaced. Meanwhile the screw is raised and the sludge pumped out, and another run is begun.

A uniform rotation of the drill and a constant watchfulness are the only means of avoiding flat or crooked holes, which are so prone to occur in conglomerate and inclined strata. Again, some of the shattered rock caves in and wedges the tool fast, from which only vigorous jarring dislodges it. If this does not do it, the accumulation around the tool may be broken or cut loose by a spear; else each piece is unscrewed, a spring-socket clamped on, and the tool raised. The experience of the drill-man and the "feel" of the jar are the only guides to the working of the drill.

If the sand-pump or reamer breaks loose it can be fished out. If grapnel does not free it, it must be chopped up or the well abandoned. A "winged" tool is often brought into play to straighten out a hole. The wings are mere projections that extend up and down some distance on the four sides of the chisel, and nearly fit the hole. A hollow cylindrical reamer is

also resorted to on occasion for the same purpose. With a vigilant, competent drill-man these should not be needed.

The progress is 10 feet a shift in magnesian limestones, more in sandstones, and less in metamorphic regions, and as much as 70 feet a day with heavy tools in soft ground. Two men constitute a shift, and about 200 lbs. of coal are consumed. The rig complete cost at St. Louis \$1500 to \$1800. The high derrick pattern is preferable to the portable plant which has everything concentrated on a truck. Many a deep hole can be drilled by the machinery, and the uses for them are numerous. There are many circumstances under which no other system can be employed than that here described. In the cement flint-rock of the Missouri zinc-fields the diamond-drill, its main competitor, was an absolute failure.

The cut on p. 401 illustrates the tools of an outfit, in order from the left: sinker-bar, wrench-bar and wrench, jars, temper-screw and gauge, small bit, large bit and rope-socket, auger-stem, and floor-circle (Fig. 213).

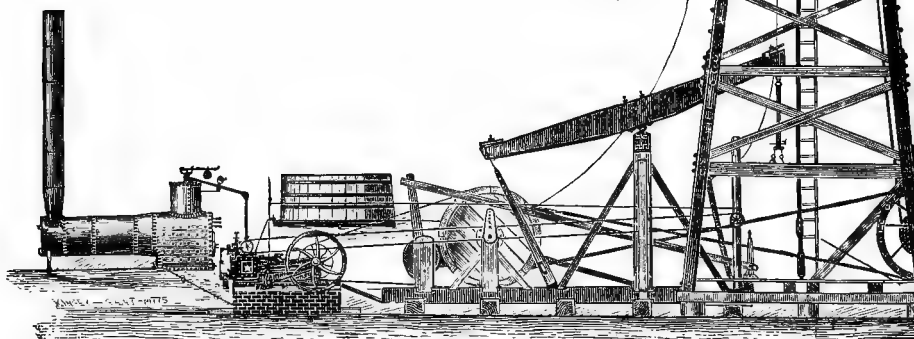


FIG. 212.

The average cost of an oil-well in Pennsylvania was \$2175
The cost of drilling wells in 1888, was, in Allegheny Co., 60

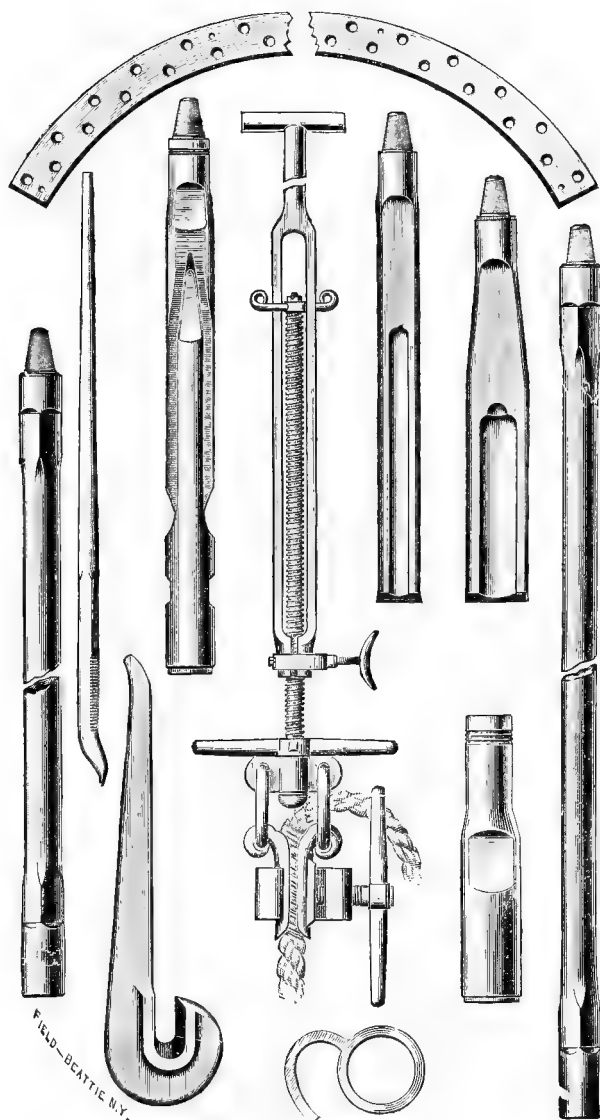


FIG. 213.

cents per foot; in Warren Co., 55 cents; and at Mt. Morris (very deep), \$1.14. Reaming is done, when required to shut off salt water, at 40 cents; use of the machine, 10 cents per foot.

Loose or fragile matter must be held up and prevented from interfering with the progress of, or jamming up, the bore by a tubing which is forced down a few feet behind the drilling. A guide-tube at the surface maintains the verticality of the tubing, and a block of wood receives the blow. The steady pressure from jack-screws is also a common mode of driving. Wood is regarded as the best tubing material, but the objections to it are apparent, so cast or wrought-iron pipe is used. In certain chemical waters zinc and bronze are preferred. The pipes may be screwed together, joined by wrought-iron bands, welded and shrunk on the shoulders, or telescoped. The first length is sharpened, driven by percussion until its full length is in the guide-tube, when another length is screwed on and driven. The hole is usually smaller than the tube. When the depth has so far increased that the pipe cannot be rammed, the bore-hole is enlarged for the same pipe to be lowered, or the same drill is used with a smaller pipe. Casing remedies another difficulty which was experienced in the old plan of boring without tubing. The water would fill up the hole and exert such a pressure (14.7 lbs. per sq. in. for every 34 ft. in depth) as would prevent oil and gas from escaping, and even flood neighboring wells.

If the drilling has been for oil, pumping for a day or two may be necessary after the proper depth has been reached. Sometimes the pressure of the confined gas which accompanies the oil is so great as to blow out the tools, and a flow follows. On other occasions communication with an oil-chamber in the sands may be opened by exploding a torpedo at the bottom of the well. Except in dry holes, this heroic treatment should not be resorted to in shaly ground, as it usually results disastrously. Formerly the shell had 6 to 30 lbs.; now 100 to 300 lbs. are used to open a cave. The torpedo is fired by dropping a cylindrical weight or a drill on it.

When the object for which the hole was drilled or lined has been attained, the tubing may be recovered by the use of an

ovoid screw-plug of oak, which is attached to the tool-rod. The plug is lowered to the bottom and a little sand is poured in to wedge it into the tube, which is now hoisted without trouble. To loosen the hold of the plug, it is only necessary to lower it and let the sand run off. A three-pronged expansible hook lowered down under the tubing may raise it. If the full column of tubing is not to be or cannot be removed, an expanding pliers is let down to cut the iron as it revolves. As a measure of courtesy to neighbors, abandoned wells should be plugged below the underground watercourse.

Often circumstances arise in which it may be desirable to isolate the several flows encountered at the different depths. For example, to shut off water from the lower petroleum, or a salt current from an artesian flow. This is readily obtained by the use of the seed-bag : outside of the tubes, at a point which is expected to dam off the flow, is a stout leather tube 6 feet long, the space between them being filled with flax-seed and both ends lashed. When the tube reaches its place the water soaks the seed, which swells and makes a tight joint, somewhat like that of the moss-box (Fig. 135).

A unique mode of drilling artesian wells is in vogue in the San Luis Valley, Colo., where 300 feet of 5-inch hole can be made in 24 hours. The soil of that valley is very porous, and for 900 feet down will hardly hold water except in the clay seams. Ditches have failed of their irrigating purposes, and each ranch is provided with one or more of these spouting wells. From a tripod a 5-inch tube is held vertically, and by a pair of blacksmith's tongs is turned by hand. At the foot of the pipe, outwards $\frac{1}{2}$ inch or so, projects a bit, which, like a carpenter's borer on rotation, will work spirally, cutting a 6-inch hole into the gravel. As fast as tubing disappears in the hole lengths are added ; and without exaggeration it may be said that in 24 hours 300 feet are run. A barrel or two of water is poured in to wash the detritus out through the ring, or a small pump is used until the ground-current is encountered a short distance down. Beyond 300 feet, horse- or engine-power will be required to turn the borer against the great friction of the pipe with the soil.

CHAPTER VI.

BREAKING GROUND.

74. Notes of cost and progress; fire-setting method; description of. 75. Description of miners' tools; the pick and varieties; underholing; shovels and spades; sledges; hammers; plug and feather; lewising; gads and moils. 76. Hand-borers; single and double hand-work; tools for the same; hammers, drills, and steel; jumpers; consumption of steel. 77. Blacksmith's work; kind of coal to be used; brief account of the materials employed in miners' tools; their selection and preparation for use; welding, hardening, and tempering, and how accomplished. 78. Varieties of bits and points for different rocks; sharpening and steeling picks, drills, etc.; making handles and helves. References.

74. THE character of the rocks which are to be excavated and the difficulties of their removal present such varied and delicate questions that it would seem useless to attempt a systematic account of the principles of and the means for breaking ground. Materials are "hard" or "soft," gauged by their resistance to abrasion as they affect drilling operations; and they are tough or brittle according to their resistance to concussion in shooting. These qualities determine the cost of breaking ground. Live, dead, or pink quartz have different degrees of hardness and toughness that experience alone can gauge. They are all hard to drill through, but the latter is so tough that only heavy charges of strong rupturing agents will make any progress. Porphyry is too variable in its constituency to have the same treatment by drill and powder over an extended area. Stratified rocks which have been metamorphosed are not affected by the same mining agencies employed in those not disturbed. The effect of a subterranean current of water, the influence of the degree of dip, and the character of the cleavage are all elements pertinent to the problem of mining, and affect the progress in

and the cost of excavation. Igneous rocks have fewer variables to affect the estimate than stratified rocks.

Estimates are easily had in any camp, but close observations of the results of a few experiments upon the given rock will supply the practical coefficients which, with the theoretical fundamenta to be discussed, should enable the engineer to determine the cost of working in the rock under examination. No criterion for general application can be offered herein; the estimate is largely speculative: yet the efficient superintendent should master this branch of practical work by which the miner may be rated.

Elaborate formulæ for estimating the cost of extraction are given in Drinker's "Tunnelling" and Foster's-Callon's "Lectures on Mining," and they are available as approximate guides; but there "cannot as yet be any rigid limits with reference to cost and progress."

A method prevailed among the ancients for conquering the hard rock which resisted their efforts of wedge and hand tools, and did great service for Hannibal in his campaign across the Alps. The miners of the Middle Ages, the Aztecs and the Japanese applied the "fire-setting" system, which is still employed on occasion as a cheap, effective method of softening the rock.

It was nothing more or less than exposing the face of the rock to fire and suddenly cooling it with water. The rock dilated, split, or was even changed in composition by this agency; and the unequal contraction on cooling further disintegrated the rock, which became amenable to the commoner tools. When we recall the instances where explosives availed little, where blacksmithing was the heaviest item of mining, one can appreciate the advantage of such a cheap, simple substitute for working the tough pink quartz of our metalliferous districts.

As practised in mines, a portable grate, inclined toward the face, on four legs, had billets of wood piled upon it, the flame and heat from which were directed against the breast by a shield overhead. In some cases the grate was of the basket

form, suspended near the roof. The wood was fired on a special day, say Saturday, and when burned out and cooled off the men attacked the calcined surface by ordinary means.

As the atmosphere is fearfully vitiated, full, free ventilation is necessary. The heat, the steam, the products of combustion of the wood, and the vapors generated by the roasted rock were such as to be intolerable, and prohibited mining during the incineration. With a powerful air-current they could be swept out rapidly, and this objection to the method overcome. But another remains to offset the economy of the work—the shattering not only of the face but of the wall rock, and especially of the roof, which would be rendered dangerous to an uncertain depth, and thus endanger lives and require prolonged re-forming of levels injured by the scaling of the material. Again, while there may be no objection to a disintegration of the rock from the action of the heat, a decomposition of the ore or mineral is another matter. It is said that in the mines of St. George the silver actually melted out of the veins, and roasted ores came from the vein for 100 feet back. So ores injured by fire cannot be subjected to this treatment. Otherwise, where fuel is cheap, ventilation good, and the rock capable of standing in large excavations this method, may be profitably applied.

75. In loose material and shattered rock shovels and spades are the only tools needed; clays and soft rock require picks, crowbars, and shovels; ground that is scaly, brittle, and seamy is managed by wedges in different ways; and massive rock cannot be broken without the aid of the hammer and drill. Where expedition is desired, these tools are replaced by machines, which duplicate their motion, with, however, greater motor power than the muscular effort of the laborer.

The ordinary shovels are made of iron plate rolled under a welding heat with an edge of steel, and ears drawn out for the handle. A concavity to the blade imparts stiffness and carrying capacity to the tool. The end is square or pointed according to uses, and set on a long or short handle. The long-handled pointed blade is the form in which they are most

used, and the "diamond-labelled" ones are the most popular. The width of the blade in inches designates the size of the shovel. The life of a shovel is greater if it is used only to lift and convey stuff. Those which are used as a pry break at the helve instead of wearing out on the bowl.

From the very nature of mineral worked, the pick is essentially the collier's tool, the hammer and drill the metal-miner's. It is variously known as a pike, mandril, slitter, hack, and mattock. It comes in different shapes and weights. There is the straight or the curved, the anchor-pick or the poll-pick; each has favor for certain work. Indeed, in the same mine several forms are to be seen; perhaps it is a matter of individual prejudice. The straight pick assists the reach, the curved enables a fairer blow. So, for underhand work, for underholing, and for getting into corners, a straight-head pick is used. In downward work the curved will strike fairer than one without any "sweep." Its curve should be, properly, one of a radius equal to the combined lengths of the arm and handle.

The length of the iron is about 22'', the largest used, to the author's knowledge, being 29'' long, which in the hands of the "box" cutters are doing remarkable "jadding" (cutting the top). The weight of picks varies between 2 and 9 lbs., with 3½ or 4 lbs. as a common weight. The heavier weights are for downward cutting.

Picks may be made at the mine or purchased ready-made, of any weight, in all steel or in iron "eyes" with or without steel ends. They may be double pointed or single, with a hammer head, called a "poll," at the other end if they are to be used for driving gads or breaking rock. The taper or chisel-pointed ends of the pike slit the rock and the handle gives leverage to pry it off. So the three desirable points of a pick, aside from the material of its manufacture, are strong cutter tips, stout eye, and a tight handle (commonly called helve by Cornishmen).

The wearing parts of the pick are the tips, which should therefore be replaceable. On this account the all-steel pick is

not advisable, for when its two points are blunted a few times they are worn so short and bluff as to be useless without having done much work. On the other hand the iron-pick eye, a 14" length of best iron, gives long service by welding on tip ends whenever desired. Kendrick's make is in favor hereabouts. In some collieries the men have picks for removable points (5 to 7 oz.), of which they carry several to work with them. The picks are sharpened to form on an anvil, and commonly drawn to a four-sided pyramidal point for hard rock, a slim taper for fissured rock, a bluff taper to cut crisp ground, and to a chisel end for chipping the ground.

Poll-picks are forged out of 1 $\frac{1}{4}$ " iron, and have a stem 12" from end to eye, an eye of about 2" long, and a stump at the other end 3" long, to form a poll for striking blows. The stem is pointed with steel, the poll faced, and its corners chamfered. The stem is long and slim for soft ground.

The eye is oval, and should be well surrounded with metal. It is formed by gashing the bar of iron in the middle, swelling it by working a drift into the cleft, and hammering out stout cheeks. All the strain of the prying falls on the eye, which must be true and stout. Many of the drifting picks have the eye raised so as to give firmer hold for the handle.

The handle is of hickory or ash, and should be selected with care. Only straight-grained, firm, well-trimmed sticks are to be accepted. The handle is trimmed to the shape of the eye into which it is hammered, and then wedged tight by a pair of iron feathers. There should not be the least wincing, nor should the feather be so keen that its driving will split the eye. Wincing could be prevented by driving a T-piece into the handle at the eye, and bending its arms under the pick. The handle should be at right angles to the pick. This may be tested by drawing arcs with the tips, and using the handle end as a centre. The radii of both arcs should be equal. The length of American handles is about 28", those of the English are 30" to 35".

This tool, properly built, dressed, and tempered, is a most

effective excavator in soft and shattered rock. It is the sole tool of the coal-hewer.

The operation of coal-mining is called undercutting, underholing, bearing-in, or kirving, and consists in cutting a groove underneath a mass of coal in the soft floor or in the coal itself to a depth of, say, four feet, after which the block is sheared by passing grooves on each side as deep as may be necessary to assist in felling the mineral held only at the back and top. There is necessarily some waste in cutting the groove, which is often 9" high at the face and 2" at the back. In some mines, for example, those working long-wall, the holing extends some distance underneath a face of coal propped during the mining by sprags (Fig. 8), and left overnight till the pressure of the superincumbent strata cracks the coal off or breaks it down. In other mines a hole or two must be drilled into the coal a few feet above the floor in order to blast the mass down. The presence of cleavage-joints materially assists the getting of the mineral; if they are close together, or if the seam parts readily at the roof, the underholing is not deep. Anthracite is rather too firm and brittle for kirving. It is blasted "off the solid."

In the operation of bearing-in the miner stands and cuts the groove a few inches deep; then he sits on the floor and carries the holing further, but ultimately lies on his side and picks the hole as deep as he can reach, meanwhile propping the mass of coal over him at every few feet. This is the most hazardous operation connected with mining. Generally the men work in pairs, and two men can underhole 25 feet in four or five hours.

The hammer and wedge has become a thing of the past for most mining work. The primitive wedges, used dry and wet, were of wood; now they are of iron, steeled iron, or solid steel. They had a hole in the side for convenience in carrying, and to insert a holder during use. Now, the wedge is used for mining out large, sound blocks of stones from rock which have a tendency to split in certain directions. The Egyptian obelisks were obtained in that way.

The plug-and-feather arrangement is much more efficient than the wedge which it has superseded, and consists of two

iron wedges, with their outer surfaces arched, inverted in the hole, while their inner surfaces are flat. They consist of a number of lengths connected by hinges. They are put together at their flat surfaces, and are introduced into the hole so that they nearly fill its deepest part, while there is some play between them at the top. A flat plug is then driven between the two pieces, acting most strongly in the deeper parts of the hole. The driving of the wedge may be done by hand or by the drill employed for drilling the hole.

Coal, slate, and ornamental stones are extracted by this means. Powder would shatter them, and start incipient fractures that would facilitate their subsequent disintegration.

Of late an hydraulic wedge has been used with good success at the collieries near Saarbruecken, the position and the effect of the driving-wedge being reversed, the thicker end being in the bottom of the hole and the edge near its top. Fitting between two half-round wedged cheeks, it is driven from below upward by the hydraulic pressure. Instead of water-pressure, the force from the explosion of powder is also utilized to force the wedge outward and snap the rock in the direction of the thin edge. This process gets out dimension-stones in some quarries, but the risk of huffing off the rock at the top of the hole is great.

Gads or moils are very useful accompaniments to a miner's equipment, differing from the early wedges only in that they have pointed tips, not chisel edges. Now they are made of a piece of steel that has done service for drilling and been dressed and smithed until it is less than 10'' long, and could no longer be used for starting a hole, and converted into amoil by tapering off the bit point to a pyramidal tip. It is used for chipping the rock, to give a smooth bearing to timbers, etc. Though the process in hard rock is slow, themoil is practically indispensable. It will do service as well to remove scaly fragments of rock. A number of them, sharpened and annealed, should always be on hand for the use of timbermen and shaftmen. Many are lost in the mining waste and débris. Themoil is used in the Lake Superior copper-mines for "blockholing" or splitting the large masses of copper.

76. The simple appliances described in the previous lecture constituted the only reliance of the early miners. With the improvement in iron and steel manufacture the methods have changed from that of slitting along the lines of cleavage to that of the actual penetration of the rock by several forms of machines. According to the character of the rock and the quality of the tool, the soft or the hard material is entered by a rotary borer or by a percussion-drill. These tools are operated by hand, or, with greater speed, by other motor power.

The variety of hand-borers presents a diversity of form, patterned after the jumper or the auger. In either case a crank is turned by hand, the motion being imparted to the borer by a more or less complex mechanism affecting the patent rather than the efficiency of the machine. Indeed, simplicity of parts is highly desirable as less wasteful of power.

Time is lost if the machine is awkward to place ; besides, the work of turning a crank is not as agreeable or as efficacious as churning a drill. Moreover, the weak spot in the appliance is in the torsion of the bit, where the strain is maximum, and of a character more difficult to resist than in any other part of the machine. Finally, the same machine cannot indiscriminately be used in rock, shale, slate, hard or bony coal. Different grades of machines would therefore be required in a mine. In consequence, the machine is heavy, wasteful in time and power, and offers no kinetic advantage over the pick or jumper, in the use of which all the muscular effort expended upon it is paid out in the intended blow. Where they were delivered to the men, the results were unsatisfactory ; for all refused to buy them, the inexperienced hand injured them, and the benefit from their introduction fell upon a few only. Various types are used in the anthracite coal-fields.

Still, Howell's and McMurtrie's drills are extensively used in large properties. The McMurtrie is more quickly set than the Howell, while, on the other hand, only one hole can be drilled from setting. It is shifted after each hole.

The percussion on the rock is accomplished on one of two

plans. The first is known as the "jumper," to distinguish it from the drill, which is the second and more common plan.

The jumper is a heavy 5' or 6' iron bar, swelling near the middle, and sharpened to an edge at each end. It churns a hole down into the rock, by being lifted at the bulge and allowed a free fall, after which it is turned slightly, raised, and dropped 1' or so. When one bit is dulled, the jumper is reversed, and the other end, a mite smaller, continues the operation. In this way a hole is "jumped down." The débris is cleaned out at intervals by a scraper or a "spoon." Only holes which are vertical can be churned, friction reducing the momentum of the drill. The holes are apt to be triangular.

Coal and limestone is churned at a rate of 40 or 50 feet of hole in a shift; granite, 15' or so.

The jumper, with one cutter and a head for hammering, has lost the significance of its name, and is properly a drill.

The cutting edge may be straight, concave or convex, and acute (slim) or "bluff." The shape is somewhat a matter of individual preference and skill. A slim bit cannot stand in hard ground, for which the obtuse convex edge is designed. The convex bit is stronger than the straight, and does not "stick" in the hole as does the concave or straight, but it is not so good a cutter.

The drill is a bar, which has one cutter edge and one hammer end. It is of round or of octagonal steel. Steel transmits the blow better than iron, and saves time and metal. The solid octagonal bars are almost universally adopted in preference to the round, for they are better turned with each blow.

The diameter of the steel depends upon how it is to be used. Where one man holds the bar and hammers it with the other hand, the size is $\frac{3}{4}$ " to 1"; for "double-hand" work, where one strikes while his companion holds and turns the drill, the diameter is from 1" to $1\frac{1}{2}$ "; for two strikers and one holding, steel as large as 2" is used. Generally speaking, it is better to have as wide a hole as convenient; but of course the single worker can only handle a small drill. In rare cases he is found working a $\frac{5}{8}$ " drill, but that is rather light and springy.

Holes over $1\frac{1}{4}$ " are also exceptional, even with a triple gang. It has been found by experience that it is cheaper to drill a narrow hole and to increase the strength of the powder than to have a hole of large diameter with a larger quantity of weak powder. The work of drilling a $1\frac{1}{4}$ " hole is nearly three times that of putting a $\frac{3}{4}$ " hole of the same length; the relative volumes of the rock pulverized are as the square of the diameters; or, in other words, all else being equal, 25 lineal feet of small holes can be put while 9 feet of the $1\frac{1}{4}$ " hole are being drilled.

Such being the facts, it is very important for the engineer to determine whether the men shall work single, double, or triple. For many reasons, mainly social, double-hand work is almost universal; yet there are not only objections to this practice, but benefits favoring the single-hand work.

In double-hand work the men alternate the work of striking and holding, and to be able to work in any sort of a position, should be capable of keeping "either hand fore,"—that is, strike right or left handed.

In narrow places, working stringers and stopes, it is highly desirable to work the men single. The miner who is proud of his work prefers single-hand, and he can be relied upon for conscientious service. It is very tedious, however, for it does not give the relief of alternating, as does double-hand. Briefly, it may be said that if the single-hand work can be enforced under capable supervision, it should be introduced, except in very hard ground. Drinker says that in "point of economy of time and money one-hand drilling is from 30 per cent in soft schists to 20 per cent in soft sandstones cheaper than two-hand drilling." In hard rock "one-hand drilling gives the more rapid advance." Dr. A. Serlo in his "*Leitfaden zur Bergbaukunde*," believes that, "except in shaft-work, all the other forms of drilling may be executed more quickly by single than double hand," and "perhaps more cheaply."

A hole is drilled by chipping the rock from the concussion of the bit which receives the blow of the hammer. After each stroke, the drill is turned about $\frac{1}{3}$ th revolution. A little water

is poured into the hole to preserve the temper of the tool and to mud the drillings, which are scraped out frequently by a spoon, or, if the hole is very wet, by a gun. A swab-stick is also much used. The hole is started wider than it is intended to be at the bottom, where it is about $\frac{1}{8}$ " wider than the cartridge. Upper holes are dry, and need no cleaning.

A spoon is a round $\frac{3}{8}$ " iron bar 40" long, with a handle at one end and the other flattened out and curved slightly for 5" or 6" of its length, then bent to form a small cup that will scoop out the débris. A gun is a syringe, made of a 4' length of gas-piping, with a suction piston and handle.

The depth of holes varies with the work, but for hand-drilling rarely exceeds 36"; single-hand work averages 25". With nitroglycerine the holes may be one third deeper than those to be fired with powder. Hard, brittle rock requires long, narrow holes; while tough or fissured material is best broken by the short, wide holes. They are also shallow, and multiplied in jointy or "vuggy" ground. While deep holes are expedient, expert miners refrain from very deep ones, except on occasion, because they may leave the ground in bad shape; and it is often as much his endeavor to secure a good bench for subsequent shots as to break ground with the present one. Drilling in hard rock is preferred to that in variable rock, which does not give round holes or a uniform wear to the steel.

Holes are drilled wet or dry, but generally the uppers are only half as fast in putting as those pointing downward, which can be kept wet; for this reason more is paid for the over hand holes than for underhand. The relative direction of holes is of no significance. There is no reason for the prejudice favoring horizontal holes, except in shaly ground, where vertical holes cave badly. Sometimes a gas-pipe or drive-tube enclosing the drill will hold up the ground till primed or even fired. About 30" of holes can be drilled single-hand, in medium rock, per shift. In quarrying limestone the holes for plug-and-feather are 7" deep, costing 29 cents per foot. Single-hand miners in Swedish iron mines

drill 5 feet per day; in very mild rock, 10 feet. An $8' \times 10'$ drift can be driven with four holes in the bottom, and three to blow down the top before squaring up. A shaft $10\frac{1}{2}'$ diameter, in medium rock, is lifted with 8 holes $30''$ deep and $2\frac{1}{2}$ sticks of giant per hole.

The consumption of steel varies; in a porphyry heading $4\frac{1}{2} \times 7$ an average of 25 lbs. per 100 lineal feet may be estimated. A double-track gangway consumes about 1.4 lbs. per foot.

Hammers are carefully selected for weight, varying from 3 to 10 lbs., according to use and preference. Single-hand men prefer the light weight, 4 or 5 lbs., and short handle; double-hand hammer-heads are provided with $20''$ to $24''$ helves. A good striking sledge is short and its weight concentrated to a large diameter. The expert prefers the round face, though the orthodox is flat. The hammer-heads used for wedge-driving are long and slender.

Different hammers should be provided for the several operations. Striking hammers should never be permitted for breaking rock. Their face is soon injured, and no miner would think of striking with a cobbing hammer.

77. The duties and work of the blacksmith may not seem relevant to the engineer, but as a matter of fact it is highly essential that he be capable of judging of the performance of the blacksmith, who comes as the intermediate between the engineer's complaints of some miner's laxity and that man's retaliation in pleading bad tools. On the other hand, a smith who can sharpen well for hard ground is held in high esteem by miners. No manager can afford to be ignorant of any element connected with the working of his diggings.

The shop should be supplied with a full kit of tools that would not cost over \$20, good bellows and tuyeres, Peter's anvil, vise, taps and dies, twist-drills, round and square $\frac{1}{2}''$ to $1\frac{1}{4}''$ bar-iron, strap and hoop iron, an assortment of carriage and machine bolts, screws, spikes, nails, a few horse-shoeing tools, benches, etc., in a space of about $14' \times 12'$, with hinge-door openings near or over the fire, in the two walls, for working long bars.

One important element of success to the blacksmith is the fuel. This may be a slightly caking coal that gives flame and a high heat. Coke is hotter, but harder to keep fire in. The fuel should be as free of sulphur as possible; white-ash coal is better than red ash; the sulphur makes the iron hot short, and tends to produce scales; the coal should be clear of shale and slate, for they fuse, and make a pasty cinder that is annoying.

A few remarks regarding the materials used may not be inappropriate. It is a positive cruelty to furnish the men bad metal, or to compel them to work with an incompetent blacksmith's product, especially if they pay for the wasted steel or for the sharpening. Wrought-iron is the most variedly useful. It is so easily worked to any purpose. Its greatest strength lies in its resistance to tension, hence it is used for straps to tie frames together; and in whatever form, each square inch of cross section is capable of a five-ton strain, or each pound weight per foot of length of bar will resist 1.5 tons of tension.

A very useful property is its capability of welding, by which two short lengths of iron may be united, to form a useful bar. The process consists of wedge-tapering an end of each bar, heating them to red, and subsequently hammering the softened parts together. A more difficult joint, known as the split, is described further on in the steeling of picks. If the welding has been well done, the point of union is as strong as any other part of the bar. The main precaution in the process is to keep the surfaces clean and free from scales, which are so apt to form in a thin fire of the forge. Scales are due to the oxidation of the iron, which while red-hot is not sufficiently surrounded by ignited carbon to consume the free oxygen of the air. When the layer of fuel is thin, or where too much blast is given, the nascent iron takes the oxygen. Once formed, the scales cannot be melted or fused off, and would interfere with perfect welding contact. The remedy against the formation, then, is to keep the iron well covered. One plan—acting as a preventive rather than as a cure—is to sprinkle borax over the surface to be fused. This slags off the

iron and keeps the surfaces bright. Sand does very much the same thing, only at a very much higher temperature. The presence of sulphur in the coal injures the welding by forming sulphide scales.

Steel is a compound of carbon with iron in varying proportions, and, though pages are written on "What is steel?" and "What steel is," all that one can say is that it occupies a chemical position between wrought and cast iron. H. M. Howe, in his "Metallurgy of Steel," says that steel, in its specific sense, is "a compound of iron possessing or capable of possessing decided hardness simultaneously with a valuable degree of toughness when hot or when cold, or both. It includes, primarily, compounds of iron combined with from, say, 0.3 to 2 per cent carbon, which can be rendered decidedly soft and tough or intensely hard by slow and rapid cooling, respectively; and, secondarily, compounds of iron with chromium, tungsten, manganese, titanium, and other elementary compounds, which, like carbon-steel, possess intense hardness with decided toughness." "This specific sense was formerly the sole one in all lands." "'Iron' and 'steel' are employed so ambiguously and inconsiderately, that it is to-day impossible to arrange all varieties under a simple classification." The various adjectives qualifying the term "spring," "shear," etc., apply to the uses to which the steel is put, and imply a certain percentage of carbon constituency.

The homogeneity of steel and the presence of carbon imparts to it a capability of hardening and tempering to a degree depending on the temperatures of the heating and the subsequent cooling. As the amount of carbon increases, the melting-point of the iron decreases; and this greater fusibility reduces its welding quality.

A steel is called "hardened" when it has been suddenly cooled, and thereby become as hard as possible. The reason for this change is not readily understood. Manifestly, it is owing to the presence of the carbon; for pure malleable iron is not in the least affected by the operation, while both steel and cast-iron are to a marked degree.

The operation consists in forging the steel to a certain temperature and then plunging it into some fluid which abstracts the heat from the tools. The quicker it is done and the greater the difference of temperature, the harder is the tool.

Either water or oil is used. Both volatilize or decompose at a temperature much below that of the immersed tools; so that the hardening takes place in a vapor formed on the principle of Leidenfrost's phenomenon of the spheroidal condition. It is supposed that perhaps decomposition takes place, whereby the hydrogen takes fire and the oxygen scales the iron. At any rate, oil contains less oxygen and hydrogen, than water, and has 77 per cent of carbon, which at the hardening temperature becomes charred. The specific heat of the oil is less than that of water, and its chilling effect is less rapid. So on the first plunge the metal is chilled and coated with soot, after which a slow process of cooling—almost an annealing—takes place. Again, instead of the iron being scaled by the oxygen of the water hardening, it is carburized by the carbon of the oil process. Finally, tests show that the tenacity of the steel is not affected in the oil as in water. Mercury and molten lead are also used for the immersion, but they are admissible only in large establishments.

Tempering is a process which follows hardening, whereby the steel is subjected to a subsequent lower heat, which softens it and removes its brittleness. To obtain the proper degree of tempering requires skill; and to attempt it without previously hardening is an exceedingly delicate performance, to be intrusted to an adept only. The risk is in overheating and scorching the metal, i.e., burning out its carbon. Iron is very easily decarburized.

When the hardened iron is slowly reheated, its surface gradually assumes phases of color, beginning with a light straw, passing through the shades of yellow, brown, purple, blue, and red. At a red heat—the original color before hardening—the effects of the chilling are practically removed.

Now, the operation of tempering consists in carrying the second heat to one of the above-mentioned colors, according

to the amount of brittleness to be annealed. This depends upon the use to which the article is to be put. As, however, it is not possible to stop the forging at exactly the temperature desired, a second stage of the operation finishes the job. The aforementioned reheat goes on a little way beyond the desired color; the article is carefully plunged part way into the water or oil, till the disappearance of the steam or fog indicates that it is cold, when another portion of the distance is further immersed for a moment. The article is withdrawn, the scales rubbed off, and the heat of the remaining portion draws to the edge, until it has assumed the proper tempering color. It is then thoroughly cooled. The impression that the steel is cooler at a blue than at a yellow, in final drawing, is erroneous; for more of the heat is conducted from the red portion to the point than is radiated to the air; the first heat to the edge gives only a yellow, which later becomes purple, then blue. Hardened drill and pick points are treated in this way, 4" of the end being heated to a yellow, and after plunging the tempering is proceeded with as above.

Caution is urged that the plunged tool, while tempering, be not held too long a time at a certain color-line, for it has a strong tendency to break there when in use. The tool should be slightly waved in the water.

Pieces which are to be tempered throughout must be allowed to "soak;" i.e., become uniformly hot before plunging.

The proper color for a given ground is only ascertained by experience. Generally speaking, the picks and drills are stopped at a straw if intended for hard rock, and carried up to a blue for mild ground. It is always desirable to preserve the toughness of the steel as far as possible; therefore select the lowest color compatible with the service to be performed. A high-carbon steel is given a lighter color than steel of low carbon.

Metal-working tools are given a pale straw-yellow; wood-working tools, a brownish tint; hatchets, saws, etc., a light purple; picks, to a rose; cold chisels, to an orange-rose; key-drifts, orange; rock-drills, yellow-orange; screw-cutting dies,

light yellow; and hammer-faces, a pale straw. A blue color would make the tools too soft for any of the above purposes.

78. A pick is made of a square iron bar $14'' \times 1\frac{1}{4}''$, heated at the middle, and then struck endwise till about $1\frac{1}{2}''$ across. This spot is softened in the fire at a red heat, cut open, and swelled by a drift, to form the eye. This—or the purchased pick-eye—is then slit at the ends and softened, while a $6''$ length of pick-steel is being heated. When ready, the steel is tongued into the iron and hammered. A reheating with borax, and a hammering, complete the weld, after which the picks are sharpened and tempered. When the job has been properly done, no signs of the weld should be visible.

"Pick-steel" is a special steel that can be had in bars $1\frac{1}{4}''$ or $1\frac{1}{2}'' \times \frac{5}{8}''$ or $\frac{3}{4}''$, and used only for tips.

Never harden a crowbar; for, its tenacity having been destroyed, it will "fly" on the application of some severe strain.

Steel bars for drills come in lengths of about $14'$ each, and from $\frac{5}{8}''$ to $2''$ diameter. The cross-section is either round or octagonal. Two brands have now equal favor—the American "Black Diamond," and the English "Jessup," which has for a long time "had the call." Our American brand is equally good, is tempered a little brighter than the Jessup, and costs 3 cents less per foot.

The bars are cut up as desired,—more economically if cut into pieces as long as can be conveniently used; $30''$ and $36''$ are the best sizes. Never cut the "starters" or short drills, for they are obtained soon enough as the long ones wear out. To save delays and be armed against emergencies, a very liberal supply of drill steel should be provided and ready for use. The bits are wider than the tool, to save weight, and also to prevent it "sticking" in the hole. They are widened according to pattern, so they can "follow" well. The first drill

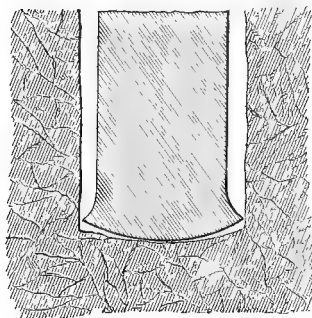


FIG. 214.

has the widest bit; the followers have narrower edges to the last one. In hard or jointy rock, where the stress increases the liability to fracture, the flare is small compared with that in soft rock, where a $\frac{3}{4}$ " drill is forged to $1\frac{1}{2}$ " wide, and to $1\frac{1}{8}$ " for conglomerate. The curved convex bit (Fig. 214) is best for ordinary hand-drilling. It is stronger, and for the increased work to be done at the circumference is more properly proportioned, than the straight.

The temper is a lighter color for hard than for soft rock, and for Jessup than for Black Diamond steel. Experience alone can tell of the proper heat. If the edges of the returned drills are cracked or broken off, the steel is too brittle, and should be softer, or other coal should be used. If the edges blunt much by wearing round, they are all right, though a harder temperature may give them longer life. Cast-steel borers are never heated above a cherry. They are annealed at the hammer end.

Generally the men are supplied with steel and tools as called for; but for many reasons it is judicious to weigh or measure the steel given out at the beginning and returned at the end of a contract, lease, or month, and to charge for it accordingly. The latter arrangement is more advantageous for the mine, if all the tools are marked privately. No kind of supervision will prevent the carelessness which buries tools in the waste or breaks handles. Loaning out the tools and holding the men accountable for their return is the only possible check.

For forging and dressing machine-bits a special set of "dollys" and "swages" are used to give the X-shape to the bits. An outfit costs \$20.

An elastic, tough wood is required for handles; and these qualities hickory and ash have. The former may be objected to on account of its weight, but it gives perfect satisfaction where used. An oval shape to the handle gives a more perfect guide to the blow than the round. They will last two years with a moderate care, but most of them meet an untimely end by breakage.

For a small mine employing 20 men in all, one blacksmith will suffice, though, of course, it depends upon what he must do. A good sharpener can dress tools for twenty men on medium rock or swage the 1 or X-bits for 7 machine-drills. Excepting the pointing of picks, the cutting of steel, and the handling of large pieces, he will need no striker. With this help he can make 12 heavy picks, 20 light ones, or weld 40 pick-stems in a shift ; or he can finish 2 sets of colliers' tools of 5 coal-picks, 2 wedges, a hammer, and 2 bottom-picks. Alone, he can dress 40 bits an hour ; with help, he can forge 25 double hand-bits, or draw out and temper 50 pick-points per hour.

Where the blacksmith does custom work for the miners, they pay \$1 per month in the coal-mines and 5 cents per bit in metalliferous mines. Usually, however, the blacksmith does the sharpening, even for the contractors, at the expense of the mine.

CHAPTER VII.

BLASTING.

79. Principles in rupturing soft mineral or rock; substitutes for powder; lime, compressed air, and wedges; theory of explosion; tables of comparative force of explosion. 80. Gunpowder, its composition; "barrel" and "needle" methods of firing; use of, and care with, powder; tools, fuse, caps; lewising; consumption of powder. 81. High explosives; nitro-glycerine, its mode of manufacture; precautions. 82. Dynamite and its modifications; composition, etc.; relative explosive effects of the nitro-glycerine compounds; their storage and care; comparative safety; tools, fuse, and caps. 83. Simultaneous firing; electricity from battery and magneto machines; difference in the caps, fuses, and care; manufacture of fulminates; relative advantage as compared with single shots; cost of electric outfit; consumption of materials; precautions. 84. Principles; direction of holes; line of least resistance; formulæ for calculating the effects of shots; influence of seams, cleats, etc.; expanding bits. References.

79. THE principle employed in rupturing rock consists in subjecting the surface of a sub-facial cavity, regular or irregular, to a sudden increase of pressure acting radially outward. When the agent is sufficiently powerful to produce a high degree of compression upon the surrounding rock, it either fractures the material by the formation of a congeries of crevices, or it shatters it. The extent of the destruction depends upon the intensity of the pressure, and the cohesion or toughness of the material.

A sudden moderate explosion may be as effective a force as a slowly applied intense pressure. In this is constituted the difference of explosives. The slow, steady hydraulic pressure obtained from the ram; that of pistons actuated by compressed air; the expansion of freezing water; the swelling of slaking

lime; and the spreading produced by wedges are illustrations of the second class of agents, all of which are, under appropriate conditions, advantageously employed. For driving preparatory workings in fiery mines, these are safer and less expensive than blasting. In foreign lands, the Coal Mine Acts prohibit the use of powder unless in conjunction with water cartridges. But they are no safeguard, since they do not entirely quench the flame of the explosives. Substitutes are sought that powder may be abolished.

Compressed air is used. A cylindrical case is inserted into the hole and connected with a powerful air-pump, that will exert a pressure of, say, 10,000 lbs. per sq. in., and breaks the coal. This is used in Wigan, is compact, handy, and, beyond the initial outlay, costs nil.

Lime has proven a safe and effective blasting agent in coal. Ordinary calcined limestone is made up into cartridges which, after insertion into the three-foot hole are tamped and moistened. In twenty minutes the expansion of the lime breaks the coal away.

The ignition of explosives is our second aid to mining which has had rapid development, as the previous agents fail to make sufficient headway in the tough massive rock. But the several means mentioned above are in vogue where the use of strong explosives may injure the material sought to be removed, where the gases from the combustion of explosives would injure the ventilation, or where the mineral is soft or seamy. Otherwise, where time is the important element sought to be gained, strong irruptive agents are employed.

An explosive, according to Andre, is a mixture "capable of being suddenly transformed into gases by the application of heat." In this sudden evolution of gas, in a space formerly occupied by a solid, a pressure is produced upon the confining surface in proportion to the volume of the evolved gas to that of the explosive. The expansive force of the gas is greater as the temperature of ignition increases. Finally, the rapidity with which the decomposition takes place is important as determining the value of the explosive. If the evolution is instantane-

ous, the maximum pressure is imparted at the moment of explosion; if the combustion is transmitted from grain to grain, its strength is dissipated over a longer period of time, and the pressure is less. Thus, the strength of an explosive is measured by its specific volume, the amount of gas it produces, the temperature and the rapidity of evolution.

Without attempting to follow the history of blasting, for which the student is referred to Rziha's "*Lehrbuch der Gesammten Tunnelbaukunst*," an enumeration of the several simple and compound substances used or suggested, at various times, to produce concussion, may be here given in chronological order: Common black powder, picric acid, gun-cotton, terchloride of nitrogen, nitro-glycerine, and ammonite. This list may seem brief, but a longer list would be merely an enumeration of the varieties obtained by the substitution of a single constituent. We have the Artillery, Sporting, and Blasting powders composed of charcoal, sulphur, and saltpetre in varying proportions; picric acid and picrates with saltpetre or chlorate of potash; gun-cotton, combined with other explosives; nitro-glycerine, with admixture of absorbents and dilutants. The result is that we have various grades of explosive compounds, from those which may be ignited by heating to a temperature of about 300° C., to the nitro-glycerine, which requires a shock. In other words, we have igniting or detonating compounds.

When an explosive is fired, the heat it develops is communicated from grain to grain with utmost rapidity and the particles decompose with the liberation of gases. According as these two phenomena follow each other slowly or quickly, we have rending or shattering powders. In the first, the gas is evolved so slowly as to give time for a concentration of pressure along a line or lines of least resistance. This is the quality desired for a sporting-powder,—ability to *project*. The slow combustion operates upon the small mass of the bullet, which can instantly take a very high velocity, and thereby give a rapidly increasing space for the evolved gases to escape. A tight bullet or plug would burst the breech or muzzle.

The miner desires to *break*, and this property is obtained

from such agents as rapidly produce gases at high initial temperature and pressure. Very little plugging is needed, for the concussion produced by the gases of the quick powder is practically instantaneous, and a wave of pressure extends in all directions, which, being resisted by the rock, spends its force there and shatters it. The more sudden the action, the more local the effect.

Powders which produce gases that afterwards either dissociate or unite, or those whose combustion is incomplete, lose some of their initial pressure. This fact is indicated by the smoke which is given off. Some of the powders give a flame which not only wastes some of the explosive force, but also is dangerous, particularly in gaseous places of fiery mines. It is estimated that fully 68 per cent of the explosive force of black powder is lost in flame and smoke.

Accordingly, the miner has the choice between slow and quick powders for brittle ground or hard and creviced rock. For tunnel-headings and sinking he needs the strongest kind of powder, the weak ones being economical in enlargements and for stopping.

Below is a table of the relative volumes and pressures of gas produced by the perfect combustion of one pound of explosive occupying about 0.016 cubic foot.

	Volume of Gas.	Heat-units.	Relative Force—	
			By Single Explosion.	By Detonation.
Blasting-powder.....	2.38 cu. ft.	900,000	1.00	4.34
Chloride of nitrogen.....	5.09 "	570,000	1.08	3.61
Gun-cotton.....	11.01 "	1,050,000	3.06	6.46
Picric acid.....	10.72 "	1,240,000	3.15	6.00
Nitro-glycerine.....	9.76 "	2,375,000	4.55	10.00

80. The term gunpowder embraces mechanical mixtures of carbon, sulphur, and salpetre, varying from 12, 8, and 80

per cent, respectively, for sporting purposes, to 20, 16, and 64 per cent for open-air blasting, and 11.5, 17.5, and 71 for hard rock under ground.

History attributes its invention to Berthold Schwartz, a St. Augustine monk, during 1320; although it is said to have been employed in Rammelsberg 200 years before. Certainly, fire-arms, fire-balls, and fiery projectiles were spoken of earlier than the 14th century.

At the present time the above-mentioned ingredients are pulverized, compressed into cake, granulated, sieved, glazed, and dried, the size of the grains depending upon the use to which the powder is to be put. It may be exploded by a fulminating powder, as well as by impact with a red-hot substance, producing, in a given case, gases in the following percentage of total volume: CO_2 , 43; N, 35; CO, 12; H, 6; carbon-hydrogens, 4,—all of which injure the ventilation.

When the hole has been put to the required depth, the powder is either poured in from a can or inserted as a cartridge, made outside. Fiery mines prohibit the use of loose powder. The powder should not fill over one third of the hole and all its irregularities, so "tamping" is necessary. In badly creviced rock or wet ground, a "clay-iron" or a "bulling-bar" often accomplishes what quick powder will not. It is an iron bar, with a ring handle or an eye at the end to permit of its withdrawal, that is used to pound clay into the crevices before loading.

Two methods of loading black powder are practised: one known as the "barrel" system, the other as the "needle." In the ordinary method the cartridge of powder is inserted in the bottom of the hole, with a needle projecting from it to daylight. Above the powder, around the needle and filling up the hole, a very soft clayey material, called tamping, is rammed gas-tight by a copper-tipped "tamping" bar. The bar may be of hard wood, but never should be of iron throughout. In many States and countries iron is forbidden by statute. The needle is replaced by a fuse, which, when ignited, fires down into the powder. The needle is a round copper bar, pointed at one end, with a handle at the other.

In the barrel method the powder cartridge is pierced by a wire which leads up through a half-inch copper tubing or "barrel" that extends the entire length of the hole. Around the barrel tamping is rammed, after which the wire is pulled out and replaced by a fuse. A comparison of the methods shows a preference in favor of the barrel, because less and a poorer quality of tamping may be used; it is twice as fast; the cartridge has less opportunity to soak water; and the cheap barrels are recovered after shooting.

Care should be taken that the tamping be free from quartz or other material that may produce sparks during the ramming. Tamping-bags of strong paper can now be had of any size, to order, at \$3.20 to \$6.00 per 1000. They may be filled with tamping of a poorer quality, are easily inserted into the hole and, by preventing contact between the tamping and the rock, obviate many risks of premature explosions from sparks.

Herewith is appended the notice issued by the Franklin (iron) Mining Co., Lake Superior:

1. That black powder and high explosives of any kind are not safe to use together in the same hole.

2. The cap is the only exploder that is safe to use in firing off high explosives, which does away entirely with tamping.

3. The only tamping necessary on high explosives is a small piece of miner's clay, which is easily put down on the charge with a wooden bar.

4. The practice of picking and boring out missed holes that have been charged with either black powder or high explosives is strictly prohibited.

5. Any miner on the mine known to use black powder and high explosives together, thereby necessitating tamping, will be discharged from the employ of this company.

The fuse, also called squib, is a thread of powder wrapped in tarred hemp, or in cotton, and waterproofed outside. The Connecticut make of fuse has the greatest favor in Colorado, and burns very uniformly. This is an important feature, for the miner may learn how short a length will give him safe escape. It burns about 20" per minute. The fuse is supplied in rolls

of 24 to 40 feet. If the tamping has been carelessly done, or contains sharp particles, the fuse may be cut and thus fail to ignite the powder. This not only delays the men, waiting ten minutes or so for the shot—if a number are fired, all the men are out, and the misfire cannot be located—but there is danger in removing the charge. Generally, it is advisable to leave the misfire alone and fire succeeding shots near it.

Though not essential, many mines have the end of the fuse in the cartridge fitted with a fulminating cap which, being a high explosive, fires the powder by detonation. This is a $\frac{5}{8}$ " copper cap, $\frac{1}{8}$ " diameter, having a small quantity of fulminate of mercury, which explodes with heat, or even by pricking, as the mutilated hands of many careless miners attest. The point of entry of the fuse into the cap is greased with a little "cartridge" soap. The California cap is popular. The XXXX is stronger than the XXX. The table in previous lecture shows emphatically the great gain of force by detonating the explosive instead of igniting it. The effect of powder is increased fourfold; that of nitro-glycerine twice.

In bituminous mines it is highly desirable and essential to break coal without adding deleterious substances to the ventilation, or elements of danger to the gases, and yet many of the primitive methods are still adhered to. Care in the handling and use of explosives is a matter of prime importance in coal mines, where the safety of the entire property of the employers and the life of the employees are mutually dependent upon one another. In metal mines the injury done by overloading or careless tamping is confined to the immediate vicinity of the victim.

For heavy galena veins, in serpentine and similar rock, and wherever the quicker shattering powders would pulverize the mineral too much, black powder is used. Granite is quarried by a procedure called "lewising." Several holes are drilled close together, and the partitions between them broken down with a flat steel bar, or broach-bit (Fig. 233). This extensive hole fixes the direction of the fracture, which is usually selected as parallel to the "rift," or cleavage. Three drill-holes make a "complex" lewis hole. The benefits of this lewising may also

be secured by the Knox system, which is meeting with favor for dimension work. The hole having been drilled, a reamer cuts V-shaped grooves in its opposite sides, to determine the line of break. The tamping is not driven down on the powder, but an air space is left between them. This scheme permits expansion of the gases and gives time to effect rupture along the plane desired.

One foot of a 1" hole can hold 5 oz. of powder, or 38", a pound. In the anthracite mines a keg of powder (25 pounds) is consumed for every 40 tons of coal mined; the bituminous miner breaks 300 tons with a keg. In Illinois the tonnage varies from 51 to 110 per keg, with a very slight difference between hand or machine work. Long-wall mining consumes comparatively no powder, which, for pillar and stall work, averages 18 per cent of the gross cost of mining.

81. Manuel Eissler, in his "Modern High Explosives," calls nitro-glycerine the "ideal of portable force," being "the most powerful known to man." Sobrero, its discoverer, called it "pyroglycerine," which, however, was so extremely dangerous as to prevent its extensive adoption. It is an effective pharmaceutical preparation for congestion of the cerebrum. Mr Nobel, seventeen years later, in 1864, discovered a means of making it safe to handle while retaining its explosive qualities. This gained for it universal adoption by imitation in every conceivable form, so that powder is fast falling into desuetude.

Nitro-glycerine, chemically known as tri-nitro-glycerine, is glonoin oil, $C_6H_8N_3O_{11}$, made by treating glycerine to nitric and sulphuric acids at a low temperature. The resulting liquid is less shattering than terchloride of nitrogen, and more explosive than powder. The temperature of its firing is 360° . It does not take fire when touched by a red-hot body, or, if it does, it burns quietly without smoke. If it is confined, it bursts its encasement. A thin layer will explode at the point it is struck. A large volume is almost certainly exploded by detonation of a neighboring body; and when explosion does take place, the combustion is instantaneous and complete.

The concussion is so rapid, that if it is laid on the face of a boulder in open air, the surrounding air cannot move aside as rapidly as the undulation of the shock, and the rock underneath is badly creviced. The air is almost as good a tamping-material as clay in powder work. The impression that nitro-glycerine works downward has no warrant in fact.

If the combustion is complete, N and CO₂ are the resulting gases, innocuous and inoffensive; if not, the lower forms of oxides are evolved, and they give rise to the objections. There should be no smoke, such as is noticed coming from the absorbents of the lower grades of nitro-glycerine. Schoen, in "*der Tunnelbau*," says "the inoffensive nature of gases of combustion has been demonstrated by experience in mining and tunnelling, even in cases where the means of ventilation are very inferior."

The liability to explosion, even by influence, is the obstacle to its more general use in the pure state. Transportation is refused it, so it must be made on the spot. For those who must use it, the following detailed account of its manufacture by Thomas Withers, of Denver, Colo., is invaluable:

"We made the nitro-glycerine in the cool stream along which our work lies—about ten three-gallon jars. The nitric and sulphuric acids are first mixed, the pure, colorless glycerine poured in, in a fine stream, the acids stirred, while they boil and send out thick red vapors of gas. If the glycerine is poured in too fast, or the water in which the jars set gets too warm, the mixture will blaze with a blowing noise, and if it is not stirred fast the blaze will shoot up some three or four feet. Keep stirring with a glass rod until action becomes less intense. Stir each jar in turn as glycerine is added, with a good current of cool water running around the jars, until the ten jars have each had some glycerine. Then commence over again at the first jar, and keep on until no more action takes place and no more fumes are given off. When all is quiet, and as much glycerine taken as the acids will convert, the nitro-glycerine will be in the bottom of the jars like a milky, heavy, oily-looking fluid.

Pour off the acids and wash with water, for nitro-glycerine is insoluble in water. An old-fashioned wooden churn with a dasher is good. After most of the acid is washed out and poured out of the jars, pour the nitro-glycerine and acids left into the churn ; dip in water, churn it up hard ; wash very clean so that litmus-paper will show no acid reaction. Keep putting in more water and churning and pouring off, for on the freedom from all acid depends the safety and keeping quality of your nitro-glycerine. When the churning is done, pour into a wooden bucket, and it is ready for use, looking milky, with a little clear water on top. After a day or two the milkiness will disappear, and the nitro-glycerine will look clear. If it grows yellowish, it shows free acid, and churn some more with cool water. If it gets orange-colored, put into the churn some lime or soda. If it begins to look deep orange and cloudy, explode it at once, or pour it out on the ground where it will not be dangerous."

82. Dynamite is a generic term that was coined by Nobel to include the mixture of nitro-glycerine with an inert or a chemical absorbent material which renders it perfectly harmless so long as it is in a state of absorption without any exudation. The compound is like moist brown sugar in color, and freezes at 46° F., when it hardens into a white mass. In this state it cannot be fired, and requires thawing out by exposure to a non-radiant heat cooler than the vaporization-point of nitro-glycerine. Immersion in a double kettle, around the jacket of which lukewarm water circulates, is the safest plan. If the kettle is kept away from direct fire or heat, no super-heating can take place.

Dynamite is almost as sensitive as nitro-glycerine to sudden rise in temperature or pressure. Atlas, dualine, forcite, etc., are but names synonymous with dynamite, differing among themselves in the material used for absorbent. Infusorial earth, sawdust, wood-pulp, and magnesia are used. Besides these inert substances, explosive bases are added to increase the strength. Nitro-glycerine is, however, the active principle, and its percentage determines the power of the explosive,

excluding consideration of the redundant chemicals. The strength of the explosive and the disruption of the rock increases with the percentage of nitro-glycerine contained in the explosive. Infusorial earth which absorbs three times its bulk of nitro, excels all other bases, and furnishes the strongest dynamite, called No. 1. The weaker grades are designated as Nos. 2, 3, etc.; No. 3c being about the lowest stock grade. No. 1 has an explosive strength of three fourths that of the pure article, and six times that of black powder, than which it is unquestionably safer to transport and store. Experiments upon the relative efficiencies of the various explosives under water have been made, and are recorded in General Abbot's "Submarine Mines;" but no formula can be prepared from the apparently conflicting results, because of the foreign substances added to the main explosive. Dynamite No. 1 showed, however, a greater intensity of action than does the "pure stuff." So do explosive gelatine, dualine, and Hercules No. 1.

The following will suggest the constituents of some explosives:

Tonite is macerated gun-cotton 52.5, and baryta nitrate 47.5.

Gelatine is soluble gun-cotton 2.5, and nitro 97.5.

Dualine is nitro 50, sawdust 30, and nitre 20.

Rendrock is nitro 40, paraffine 7, nitre 40, and wood fibre 13.

Atlas A is nitro 75, fibreless wood 21, nitre 2, and magnesia 2.

Hercules No. 1 is nitro 75, chlorate of potash 1, nitre 2, sugar 2, magnesia 20.

Giant No. 2 is nitro 40, rosin 6, sulphur 6, absorbent 8, nitre 40.

Rackarock is nitro-benzol 22.3, chlorate potash 77.7.

Vulcanite is mealed gunpowder and nitro in different proportions.

The storage of nitro and its compounds should be in large, dry, airy caves or earth-covered sheds provided with ventilating flues and open gratings, but no solid doors. It is dangerous to permit dynamite to become wet, for water will replace the nitro by capillarity. Some of its forms are very sensitive to a sudden rise of temperature, and all to a slow heating which decomposes them. When it shows an acid reaction, or the least signs of deterioration, it is liable to spontaneous explosion, especially if strongly confined. But if it has been properly made, combustion cannot ensue, at least not for several years.

The French Government reports only four accidents, with no deaths, during eleven years, per 100 tons of stored giant as against 7.5 with an equal storage of powder.

Dynamite should never be stored with caps; nor should cartridges be laid away with the caps attached.

Not only are the nitro compounds less dangerous than powder to handle, but the work of the miner is lightened. The tamping is less, and, indeed, No. 1 requires none whatever; the mineral is broken finer, requiring less dressing at the surface; smaller holes can be drilled; the consumption of steel and supplies is less; and the mining time is materially shortened. Another important advantage is, that water does not injure it, hence it is incomparable for wet ground. As the harder rocks are encountered, the advantages become more manifest. If a rock is so hard that small holes and dynamite cannot make much headway in it, the only hope is to "fire-set" it first.

Holes are loaded with nitro by pouring with a tin cup over and upon the fuse and cap, and covering with water. In creviced rock a bottom is made of sand or dry earth before the charge is made. In a narrow crack the explosive is allowed to soak in and be fired by the influence of others.

The loading with dynamite is identical with black powder. The giant, coming in cartridges to fit the hole, is placed, as many as desired, in the hole, and with a safety fuse or electric wire and cap slightly tamped.

Miners sometimes use black powder with giant; "it starts the hole." The fact that the entire ground to the very bottom of the hole is broken,—i.e., no "collar" or "bull-ring" is left,—is sufficient evidence of its advantage. This is practised everywhere in the proportion of $\frac{1}{2}$ pound of black to three sticks of giant.

The great disadvantage attending the use of any form of explosive, particularly in coal-mines, lies in the production of smoke or gases more or less noxious. The combustion of carbon, hydrogen, and sulphur in the presence of the oxygen provided by the various nitrates added to the explosive, results in the production of carbonic acid, carbonic oxide,

sulphurous acid, nitrous and other injurious gases, and water, which necessitate plentiful ventilation. When the combustion is perfect and complete, all the carbon is converted to carbonic acid, and the sulphur to sulphuric acid; but in the deficiency of oxygen by the use of adulterants or due to an imperfect mechanical mixture, carbonic oxide is developed as well as smoke, and with the other injurious and combustible gases, is ejected from the hole into the room or stope. Sometimes this is accompanied with a large volume of sparks, but always with the emission of noxious products of combustion which are dangerous to health; while such as are inflammable are subject to secondary explosion, and the ejection of an invisible volume of heated gases is just as dangerous as the actual flame. In rooms or workings holding in suspension a little dust, the combustion of this hot gaseous product of the explosion which is projected into the air is there completed, being aggravated by the presence of the supplementary combustible coal-dust, and may be followed by an explosion, for an elongated flame is produced which may be carried for a great distance by the burning of the soot. This flame may be the nucleus for a bigger explosion, when brought into contact with a neighboring volume of gas.

Common black powder, blasting gelatine, and carbonite produce carbonic oxide in larger quantities, and these explosives should be prohibited from use in any coal-mine—certainly from dusty and fiery mines. Not only are these explosive gases deleterious to health, provocative of flame and explosion, but they are wasteful of energy. Compared with them the high explosives nitro-glycerine, or dynamite, are eminently safer and more economical. The Committee on Explosives in Coal-mines concluded that the use of high explosives would greatly reduce the risk of explosion in dry, dusty, and fiery mines. The combustion of the elements of high explosives is complete and instantaneous. The total energy is converted into heat, and rupture of the rock ensues at once without expelling into the atmosphere much waste heat, while with the former the imperfect combustion of the

elements results in the formation of a gas which is exploded ultimately in the room, producing disaster, not useful results. Black powder is a mechanical mixture, while the dynamites, a chemical mixture, is capable of more ready decomposition. Every commission which has been appointed and examined into the safety and efficiency of blasting-powder has reported the one same fact—that it is highly dangerous; in many countries it is absolutely prohibited. In December, 1896, an order was issued by the Home Secretary (Great Britain) scheduling a list of explosives permitted to be used in coal-mines, and another of those absolutely prohibited. Black powder is not entered on the first list. He, however, names a variety of flameless compounds which may be termed safety explosives, and are now known as Sprengel explosives. These flameless compounds consist of a mixture of two solids, a solid and a liquid, or two liquids, one of which should be hydrocarbon, and the other a compound rich in oxygen, neither, at the same time, being sensitive to friction. In 1886 a group of explosives was introduced, principal among which were ammonite, roburite, carbo-dynamite, and bellite, in which the chemical to supply the oxygen is a nitrate, as in black powder. Nitrate of ammonia or of barium was most commonly employed, while the other combustible is nitro-naphthaline or dinitro-benzol. Owing to the deliquescent nature of ammonium nitrate the two elements are usually not mixed till ready for immediate use, and when finished the explosive must be kept out of contact with the atmosphere by dipping it in melted wax. Changes of temperature do not affect the mixture; freezing for three months does not injure it, and there is no exudation to endanger it. Unlike gunpowder and all nitro-glycerine compounds, a mixture of this variety will not explode by percussion, fire, or electric spark. If struck with a heavy hammer, the portion of the explosive directly hit is decomposed by the heat developed by the blow, but the remainder is not affected. If mixed with gunpowder and fired, the latter explodes, scattering the former without affecting it.

A detonating wave such as is produced by the fulmination

of mercury is alone capable of starting an explosion. When the two elements are intimately mixed in proper proportions, the shock of the detonation is communicated to the layers of molecules in the immediate proximity of the cap and the powder, whereby the "molecular edifice" is destroyed. This initial force is augmented to the degree corresponding to the heat evolved in the decomposition until the total is consumed. The decomposition of the ammonia nitrate absorbs such an amount of heat from that of the initial force as to reduce the final temperature of the gases developed and projected into the air below the temperature of the ignition of fire-damp. Owing to the possibility of an incomplete combustion, an excess of available oxygen is added in an excess of nitrate of ammonia, which, absorbing more heat, effectually prevents the formation of carbonic oxide and nitrogen oxides. Deflagration, however, frequently results with all its attendant dangers, the spluttering stream of sparks being almost certain to ignite the dry coal-dust which may be present in the room. This new source of danger arises from the use of the detonator, the strength of which requisite for exploding the blast increases with the inertness of the two constituents. In the report just referred to it was ordered that ammonite containing between 87 and 89 parts of ammonium nitrate and 11 to 13 parts of di-nitro-naphthalene should be fired by means only of a detonator or an electric detonator fuse containing not less than 23 grains of fulminate; so also for bellite No. 1, which contains from 79 to 81 parts of nitrate with 19 to 21 parts of meta-di-nitro-benzol. Roburite, with nearly the same amount of nitrate but containing the more sensitive chloro-naphthalene and di-nitro-benzol, was ordered to be fired hereafter by a detonator No. 6, containing 15 grains of fulminate. Carbonites containing 30 to 36 parts of barium nitrate, 25 to 27 parts of nitro-glycerine, and 37 to 43 parts of wood-meal also require a No. 6 detonator. Ardeer powder, containing a little more nitro-glycerine than carbonite, with 11 to 13 parts of kieselguhr, require for its explosion a No. 3 detonator containing 8 grains of fulminate. Though

the use of these detonators adds somewhat to the danger by their deflagration, nevertheless the permitted explosives are far more safe than the other powders now in use. The use of a higher-power detonator will result in a total combustion which would then reduce their danger, but the heat evolved may be sufficient to ignite explosive mixtures of gas and air. The strength of the detonator, therefore, is that which is appropriate for the given mixture, but it is doubtful if any rupturing agent in which complete detonation takes place can be fired without generating a degree of heat that is likely to ignite an explosive mixture.

The expense for safety explosives in mining coal is twice as great as that for black powder. With ammonite and carbonite, as compared with black powder, there is an increase in the proportion of coal-dust formed. Experience shows, also, that in brushing the sides are left more ragged with permitted explosives, and require more timbering.

In mines which by reason of their dry and dusty nature cannot be endangered by the use of even the "safety explosives," lime-cartridges may be employed with security. Water-cartridges, however, are not commended.

Powders which are said to be smokeless are made by mixing, in suitable proportions, gun-cotton and nitro-glycerine, their combinations being effected by the use of camphor and acetons. Their energy is moderated by the addition of inert materials or by increasing the proportion of gun-cotton. These all, however, reduce the rapidity of explosion. Such powders are handled in small cubical grains, and are safe. For example, nitrated gelatine (10 per cent of gun-cotton, 86 of nitro-glycerine, and 4 of camphor) can be stored a long time without fear of any chemical change. It is translucent and elastic, and explodes by detonation, not by percussion.

W. Wallace recently conducted a set of experiments upon the strength of various explosives with the following results:

Nitro-glycerine.....	100.00
Nobel's smokeless powder.....	92.38
Explosive gelatine (made from nitro).....	88.95
Gun-cotton (U. S. naval torpedo-station).....	83.15

Dynamite, No. 1.....	81.31
Tonite... ..	68.24
Rackarock.....	61.71
Atlas powder	60.43
Ammonite.....	60.25
Melinite.....	50.82

83. The fullest benefit of these high explosives can only be obtained by the use of strong detonators, and by arranging a number of shots in such positions as to be fired simultaneously and mutually assist one another. As has been noted, the explosive effect of any powder progresses radially in every direction, but the break is toward the free face, and it follows that power is lost at the back or in the solid ground. The amount lost increases with the quickness of the explosive. If, however, several contiguous holes be so placed that their backward rupturing tendencies may superpose they will shatter the rock which would, by single shooting, have been only fractured.

The zone of explosive effect may be divided into three sections: a sphere of pulverization immediately surrounding the bottom of the hole; a cone of rupture with its apex near the centre of gravity of the cartridge, and its base of an area dependent upon the relative explosive and resisting powers; and lines of fracture that extend into the rock.

The effect of synchronous firing is about 1.4 times as great as that obtained for the same amount of powder fired in consecutive shots. Benjamin Frost says in his report on the Hoosac Tunnel that "greater depths of holes are admissible," and greater advance attained.

The sole means of obtaining perfect simultaneity is by electricity; fuses will not burn with sufficient regularity to rely upon them. The holes are charged, the cap is inserted, two wires take the place of fuse, and the tamping is done. The wires are separated, each one being connected with its neighbor in such manner (Fig. 188) that a continuous wire circuit, *E*, extends throughout the series of holes. *A* is the fuse, *B* the cartridge, and *C* the cap.

The men drill the holes till nearly "tally" time, the loading and connecting of them being effected by the foreman, who is

the last man to retire. When he has assurance that all hands are out, he makes the final connections, raises the handle, *F*, and with one movement fires all the shots.

It may be well to state here that electric firing is equally applicable to black powder, indeed, a greater gain in force is

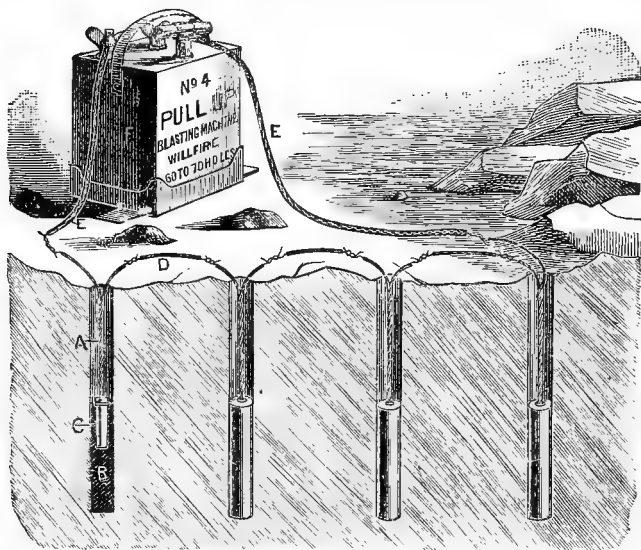


FIG. 215.

had from its detonation than from that of high-grade explosives (see table, p. 426).

In addition to the increased efficiency, other important advantages are secured. The intervals of long waits of men who stop to fire off a couple of shots at any time during mid-shift, and drive all of their neighboring comrades to shelter, are obviated by firing just before quitting-time. The smoke can then clear off between the shifts instead of vitiating the atmosphere while the men are at work. It fires with certainty and eliminates the numerous causes to which premature and slow explosions are due, and it enables firing at a definite, safe distance. On the whole it should be encouraged wherever practicable

There are two classes of machines used: a dynamo or magneto-electric machine (Figs. 216 and 215), producing medium tension electricity, and employing platinum fuses (Fig. 217); or a frictional machine which gives very high tension electricity to ignite gold-leaf fuses. The magneto machine is safer, cheaper, and more reliable. It can be had of a size to fire from 10 to 70 holes, irrespective of their distances

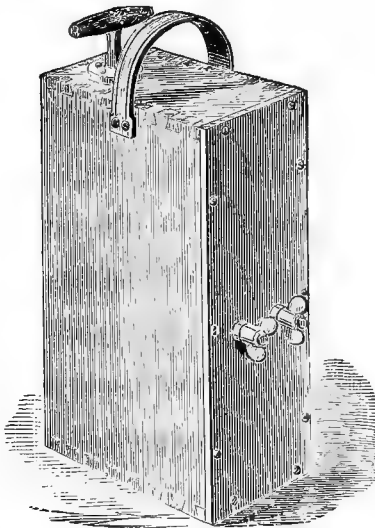


FIG. 216.



FIG. 217.

apart—within the moderate working distances that a mine affords.

The fuses cannot be indiscriminately used by either machine. The wires *C*, Fig. 217, terminate at *E*, in a fine thread of platinum, which, upon the passage of the electric current, becomes incandescent, and ignites the fulminate, *B*, of the cap. High-tension fuses are waterproof, and must have perfect insulation. The low-tension fuse may even be connected to a bare wire. Beside this advantage, its resistance may be tested by a weak current. The early defects in the priming of electric fuses, which rendered them uncertain,

have been remedied. Those now on the market have a uniform and standard resistance.

The fuse should be used with wires attached to it long enough to protrude from the surface after the hole is charged. A fuse with 6-foot wires costs only \$3.50 per 100; with 14-foot wires \$5.60. Common safety-fuse costs 40 cents per 100 feet.

Fig. 218 shows a section of the magneto machine, which consists of a weak principal magnet, *A*, with an armature, *B*, revolving between its poles; said armature is rotated by the rack-and-pinion, *C*, operated from the handle on the down stroke. As the rack-bar strikes the spring, *D*, at the bottom, it breaks the continuous current between the two platinum bearings, *E*, and causes it to pass through the outside circuit, the fuses, etc. A machine which will fire 16 shots at once sells at \$25; one good for 70 (Fig. 215) can be bought for \$55.

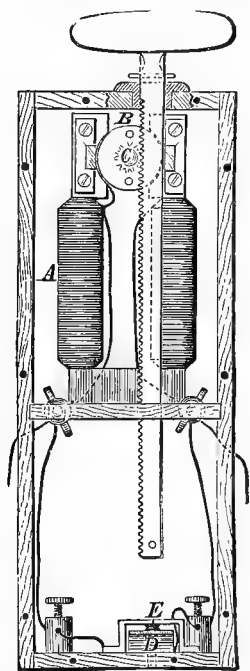


FIG. 218.

Between the machine and the fuses is a pair of well-insulated "leading" wires, *E*, Fig. 188. They are usually wound on a pair of reels, to pay out with the progress of the work, and cost \$1.25 for the reel, and a cent a foot for wire.

At *D*, Fig. 215, is represented "connecting-wire," a common wire for uniting the fuses, bought for 10 cents per lb. Some of this is lost, though much of the connecting- and fuse-wire can be regained after shooting. In a mine firing 80 shots at each "tally," the consumption of connecting-wire averages nearly 2.4 lbs. per 24 hours; in a small, double-track mine tunnel 2.6 oz. of connecting-wire, and 7 exploders per lineal foot; in the Musconetcong Tunnel, 0.32 foot of connecting-, 0.066 foot leading-wire, and 0.7 exploder per cubic yard of rock removed.

The mine may be divided into shooting-districts, and each

successively fired by the same man and machine at the level mouth; or the several districts may be coupled by permanent lengths of leading-wire, and placed on the same circuit.

The precautions to be observed are: (1) that the wires are not broken or in contact with wet rock; (2) that the boss or blaster shall touch his hands to the wet rock before working with the wires; (3) that the wires at the point of union present a fresh, clean surface; (4) that the wires are tightly twisted together (Fig. 215). This is *essential* with the magneto, and expedient with frictional machine; (5) that the advertised limit of the machine be not exceeded; (6) that connections with *E* (Fig. 215) be not made until the moment of firing; (7) that, where the battery is used, all its latent or engendered electricity be let off by touching the wires together before connecting.

84. In the manifold operations of mining, tunnelling, etc., the circumstances of drilling and blasting are so varied that empirical rules cannot be formulated for general use. The character of the material to be excavated, the area and openness of the face of attack, the prescribed limits within which the miner works, and the difference in treatment to be given vein or country rock, are a few of the contingencies that influence the efficiency of this item of underground expenses. Only experience can premise the effect of the explosive, for it becomes almost a matter of instinct instead of rule. Nevertheless, certain fundamental principles may be cited as guides taken from the experience of military engineers, who are our only source of systematic information upon this point.

When an explosive is fired, the tension of its gases acts in all directions upon the confining rock. Where the resistance is least, a tendency to rupture takes place. With powder the gases find time to concentrate their pressure upon the line to the nearest external point, and, perhaps, may break off a cone of rock. The high explosive, on the other hand, is so instantaneous that a concentration of force is not effected, and the rock will break anywhere as soon as at the weakest line.

The placing of holes should have due regard to the structure of the rock. Whether of igneous or aqueous origin, it is

traversed by a congeries or by systems of planes which rive the rock into more or less regular blocks. These rifts, cleats, or seams constitute the lines of slight resistance which are advantageously employed as lines of rupture. The quarrymen endeavor to select them on which to split the stratified rock by lewising or wedging. In the work of removing the more massive rocks, even in a fractured condition, the crevices or free faces may avail.

Under no condition can the miner arbitrarily select the lines of fracture, except it be in "tight" ground and massive rock. He then depends upon the shortest vent from the explosive chamber to the surface, and is guided thereby. Any clay seam, gouge, or fault is hailed as a welcome accessory. By "tight" ground is understood that which presents only one free face,—containing no protuberances or cavities.

In each case the line of least resistance is the objective. The shorter it is, or the more brittle the material, the less or the weaker may be the disruptive agent. To secure this with the minimum of drilling requires the instinct which experience imparts, but a few rules may assist the judgment.

A hole should not be on the line of least resistance, though it may be in its plane. If a known gouge, fault, or crevice traverse the rock, the direction of the hole should be normal to that plane, and for the reason that the explosive will find vent along it the hole need not be carried down to the seam. The hole will break to *a*, Fig. 219, if the powder is disposed wholly within the bottom layer. Sandstones and limestones are

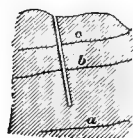


FIG. 219.

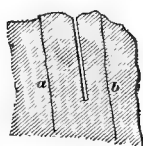


FIG. 220.

better split by weak powder; a strong one may pulverize a large portion of it without breaking stone. The roof holes are usually fired first in driving a face through strata pitching toward the men (Fig. 199); the bottom holes pre-

cede the uppers in a reverse pitch. With sufficient explosive the amount of material removed is measured by the entire block of stone from the face to the plane of the holes (Fig. 220).

Colliers avail themselves of the cleat in coal, which comes away with great freedom if properly attacked. Generally it has one direction only, sometimes two, producing cubical or rhombohedral coal. In flat seams the trend of the main cleat determines the direction of the attacking breasts and, correspondingly, the order of mining. The most important galleries are run with the cleat, and the headings perpendicular to it are called butts. In pitching seams the dip is of more importance than the cleat.

Homogeneous rock, and particularly massive rock, has no crevice or seam to assist the miner to its displacement, and an additional element is added to his work. He must consider not only the volume of the rock to be removed, but also the state in which the shot will leave the face. In other words, each blast is a "bearing-in" shot for the next succeeding (Fig. 221). In stratified rock or seamy ground the shooting is to the joints, and the stone breaks well, just as if the seams are open faces. Porphyry and quartz is always "tight" ground, i.e., there is no seam to shoot to; there is only one natural face for attack. The careful and experienced miner will see to break to benches which will offer favorable opportunity to displace large masses with little powder.

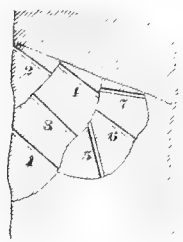


FIG. 221.

Where simultaneous firing is practised upon several neighboring holes, less heed is paid to this matter, for one cannot foretell the shape or volume of the cavities opened. Generally, several holes looking toward one another are fired merely as bearing-in holes to facilitate blasting. For this same reason, the procedure by machine is entirely different from that by hand; all the required number of holes decided upon are drilled in one heat before any blasting is attempted, and it does not signify if a hole or two too many is drilled; so the cost of dynamite is naturally higher than by hand, of steel consumed, more, and of labor, much less.

Manifestly, with single holes, the miner must drill the hole

with some reservation as to future needs, and so place it as to accomplish as much as possible with the explosive. The two forces to be considered—the strength of the powder and the resistance of the rock—may be known, the first, accurately, the second, varying with the cohesion of the rock approximately.

The drilling resistance is not the same as the shooting resistance. Trap, granite, and syenite are firm and brittle; they are hard drilling but easy shooting. Pink quartz neither drills nor shoots well. Dolomite, amygdaloid, limestone, and porphyry drill easy, but break short. In other words, the components of the rock may be hard, but if the grain is open it is not difficult to work. Drinker's "Explosive Compounds" gives a table of relative resistances of different materials and the coefficients of their toughness. Having, besides, the coefficient of the rupturing effect of the explosive upon a certain material, the excavation may be ascertained from the formula,

$$W = CL^3,$$

in which W is the weight in ounces of the disrupter; L , the distance to the face in feet; and C , the charging coefficient dependent upon the rock. In a given mine, the value of C may be experimentally evaluated by repeated trial. And the rational loads in any other case are thus fixed with a moderate degree of accuracy. Thus, if a 27-inch hole shows an average of 0.5 oz. of dynamite No. 1, C is 0.38. A subsequent 40-inch hole, under like conditions of rock and agent, will require 3.3 oz. The volume of rock thrown is estimated to be approximately equal to the cube of the line of least resistance, though it will be greater with several open faces. Against one face only a shot breaks out a funnel approximately conical.

The relative position of the line of least resistance varies somewhat with the position of the hole and the condition of the face. It is the line of general throw and rupture, and extends from slightly below the centre of the explosive to the nearest external point (line ab , Figs. 222 to 225), measured perpendicular to the free face or to the direction of the hole.

In soft rock, with a moderately slow explosive, the line may be quite long comparatively, but the same powder in tough rock cannot get far away from the face, and ab is small;

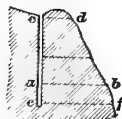


FIG. 222.

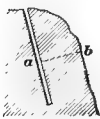


FIG. 223.

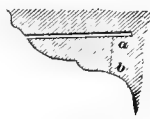


FIG. 224.

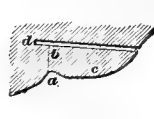


FIG. 225.

while with medium rock it may be three-quarters the depth of the hole.

If the hole be placed as shown in Fig. 226, the hole becomes the line of least resistance, and a "pop" shot



FIG. 226.

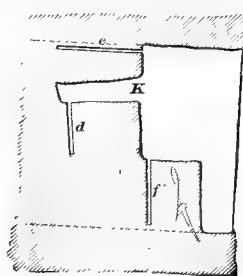


FIG. 227.

results, no matter what the rock. So, too, d , Fig. 227, fails to break; e has a very short line to break, while f is about right, in the average rock.

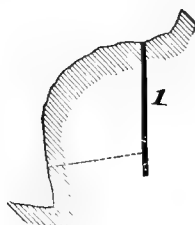


FIG. 228.

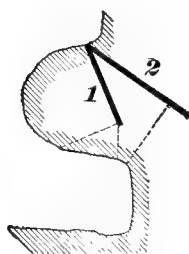


FIG. 229.

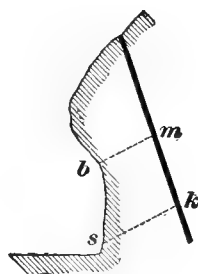


FIG. 230.

With common powder the holes cannot exceed an angle of 45° with the flush face. With dynamite, 60° is a limiting angle for almost any variety of rock. A larger angle is advised

only when a free face offers a hollow or bunch (Figs. 228 and 229). Such an exigency, while it may require a deeper or a shallower hole than that of the average hand work, increases the efficiency of the blast. In Fig. 229, hole 2 will displace more than hole 1 with equal powder and work. Fig. 203 illustrates an unfavorable hole. If very deep it will blast out to *ks*.

A hole, *om*, will do proportionately better for the same

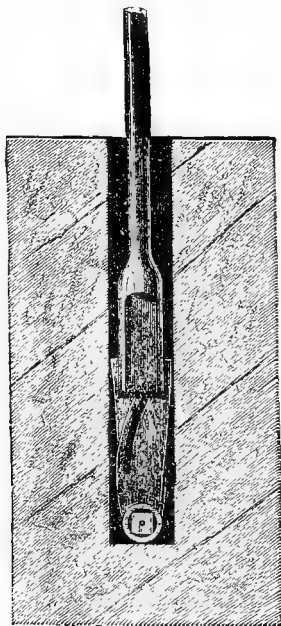


FIG. 231.

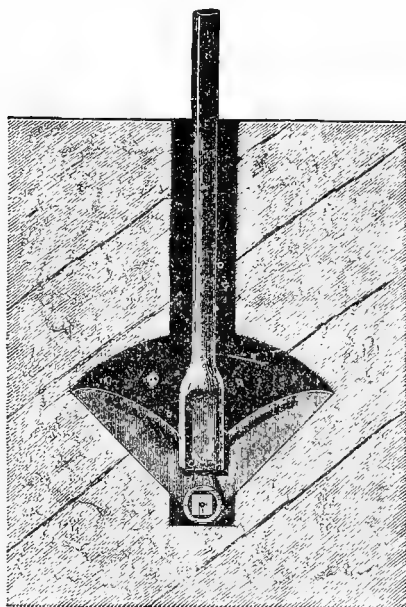


FIG. 232.

weight of powder and much less drilling; but then the subsequent removal of the block of ground *bmsk* will require nearly as much powder as for the original hole, *ok*, the line of resistance being the same.

Blasting in homogeneous material is more satisfactory than in short fissured rock, which can only be worked with shallow holes. It is also true that drilling uniform rock, even if hard, is preferable to putting holes in variable rock. In large galena

veins, deep, narrow holes do great execution; so "squibbed" holes in the flint-zinc-lead beds of Missouri. The direction of the holes relatively to the earth has no influence pro or con. Vertical or horizontal holes are equally effective, other things being equal, except that in shelly ground horizontal holes are preferred because of smaller liability to caving in.

Occasion arises when it is desirable to have deep holes—the ground may be brittle and coarse-fissured; or, occasionally, a deep hole may be desired in hard rock, and sufficient powder to do the work cannot be crowded down into the hole. In such event a chamber is prepared by exploding a light charge of giant under heavy tamping. Into the cavity thus created is tamped ample explosive for the purpose. This process is called "squibbing."

Expanding bits are also used to accomplish the same purpose (Fig. 231). When the desired depth has been reached, a pair of cutter-wings are forced out (Fig. 232), and in rotating cut out a hemispherical chamber. In soluble rock, acid poured into the hole will eat away a space for the powder.

When a streak of rich and brittle, or soft, mineral is to be recovered from an extensive exposure, it is blasted separately from the rock, which has a different degree of tenacity. Long lightly loaded holes are drilled in or alongside of the ore. On this account hand-work is more economical than machines in mines of high-grade thin ore streaks. In shafting it is seldom that any attention is paid to the reservation of ore, so that the methods more commonly adopted are similar to those used in driving. In a shaft long in proportion to its width, two centre-cut craters are separately fired.

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CHAPTER VIII.

DRILLS AND DRILLING.

85. Channellers and quarrying machines ; cost, economy, and use ; tools needed ; steam and pneumatic power. 86. Percussion drills ; requisites for a good drill ; construction ; valves and improvements ; descriptions of the different drills in the market, Rand, Sergeant, Ingersoll, Burleigh, Schram, and Darlington. 87. Rate and length of stroke in hard and soft rock ; drifting, sinking, and stoping by machine ; relative cost and progress by machine and hand labor ; shapes of bits, tools, connections ; column *vs.* tripod. 88. Diamond-drill ; description of machine ; operation ; gear and hydraulic feed ; solid and annular bits ; consumption of stones. 89. Rate of progress ; economy, cost ; its function as a prospector ; mode of keeping its record ; Brandt's drill ; electric drills ; perforators and entry machines. 90. Size and depth of holes ; system of arranging holes ; Mt. Ceniz and St. Gothard system ; the American "centre-cut" system. 91. Brain's radial system ; progress, cost, and ratio of cubic foot broken to the foot of hole ; Gen. Pleasant's method of long hole or continuous drilling by diamond drill. 92. Coal-cutting machines ; discussion of the types ; comparison of the work done, with hand-labor ; account of the Harrison, Jeffry, Sergeant, Lincke, Winstanley, Marshall, and Frith's machines ; electric cutters. **References.**

85. THE successful substitution of machinery for hand-labor has proved a most important advance in engineering. The extraction of fuel, ore, and rock is more economically and rapidly accomplished with greater comfort and safety to laborers ; hard rock is no longer an obstacle, and very long and large tunnels are rendered possible. The time spent on preparatory workings is shortened, and this element of time is an important consideration in the rapid opening of, and quick returns from, mines. As machinery never "strikes for wages

or time," irregularities and "shut-downs" are less frequent than formerly.

Every form of hand-labor tool has been successfully imitated and extensively introduced. The quarry methods of lewising (p. 429), "jumper" (p. 411), saw, chisel, pick and auger, find their counterparts in the channeller, percussion-drill, coal-cutter, and diamond-drill.

In days of yore the quarrying of dimension-stone was accomplished by the trenching along lines decided upon. Carried often to 10' depth and wide enough for a man to operate his pick, these trenches wasted much good material. The channellers and gadders now used dig these trenches as deep as desired, but only 2" or 3" wide. These machines are mounted in different styles, and cut perfectly true lines at any angle with or across the strata.

For extensive quarries these machines are mounted on a portable sliding carriage, with boiler, rails, etc., and a feed which automatically moves it with the progress of its channel. A set (gang) of five cutters receives a reciprocating motion from a steam-piston, through a connecting-rod, or through some yielding contrivance from the crosshead of the engine. The latter gives an elastic blow to the cutters. Automatic contrivances keep the cutters to their work. Machines are also supplied for cutting two channels at a desired distance apart; these are known as "double-gang machines," and cost from \$1200 to \$2000 complete. With 3 men and 400 lbs. of coal, at 150 strokes per minute, they cut from 75 to 400 sq. ft. of stone, —the **former** in marble, the latter in soft lime,—and replace 50 men.

Many quarries employ, instead, a steam or air drill, mounted on and traversing longitudinally a long stout bar, which lines up the work. This frame is comparatively light, and is adjustable to a high or low position and for vertical or horizontal holes (Fig. 233). With this a channel is cut to the length and depth desired; or an X-bit drills round holes, at certain distances apart, to full depth, the partitions between to be broken down by a broaching-bit (Fig. 233), or shallow holes are drilled for plug and

feathers. 300 linear feet of 2' holes are "put" in 10 hours, or 70 sq. ft. of channel; in granite, 28 sq. ft. of channelling is done. The U. S. Census Reports show the cost and progress in quarrying to be very varied, with a marked improvement over hand-labor in both respects. Moreover, the value of the stone is enhanced, being less shattered, as also the value of the quarry, because all the stone is saved. In Vermont the Ingersoll percussion and the Sullivan diamond-drill are used.

A tripod can be had arranged with a slot movement to the

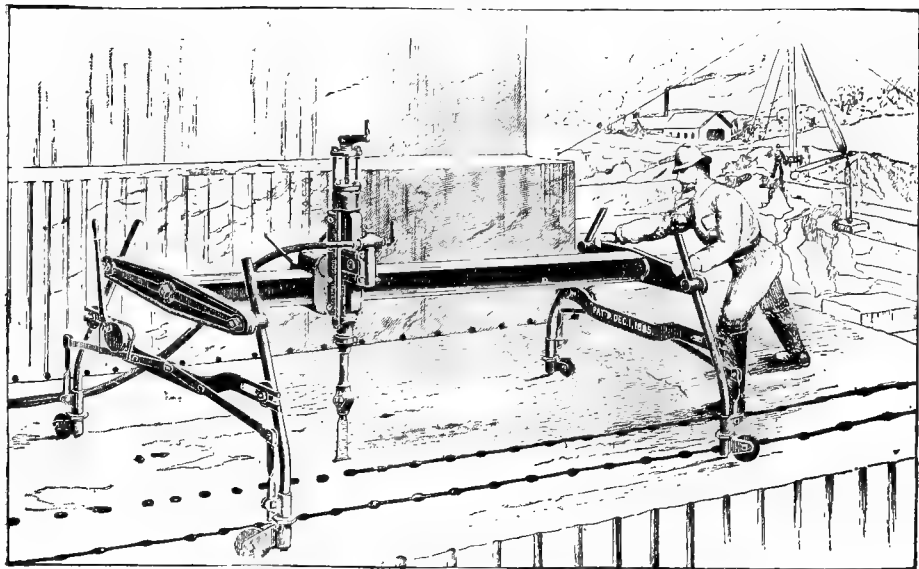


FIG. 233.

drill body, so that, with one setting, three parallel holes can be drilled for "complex lewising."

In limestone quarrying 5-foot beds, 23 holes of 7" depth can be done per day by hand, and 400 by machine. In blue-lime, steam-power drills are seven times as rapid and one-fifth as cheap as hand-work.

86. Power-drills depend upon percussion for penetration of the rock. A steam-cylinder, sliding in a guide bed-plate, mounted on a tripod or column, and a cutting-tool clamped

as an extension of the piston-rod, comprises the mechanism, which has attained a simplicity of parts that has made it the "chief element of mining success." C. D. Lawton, Commissioner of Mineral Statistics of Michigan, says that "in the progress of Lake Superior mining two forces must be allowed to have the precedence before all others—the air-drill and giant-powder."

Its comparatively small weight, 200 to 350 lbs., makes it portable, and yet it has enough metal to withstand the extremely hard usage it must receive. It occupies a small space, and can be set up in a stope or room without greatly interfering with the removal of broken rock, and will drill holes in any position or direction.

Steam is the motor fluid above ground, and compressed air below, with an ordinary pressure of 50 to 80 lbs. per sq. in. The horse-power of the drill is estimated as a simple steam-engine, with the important difference that the ratio of the area of piston-rod to piston is larger. Again, the steam-engine does its work throughout the entire stroke, but the drill-engine only at the end of its stroke. Hence it can never work expansively. The air enters the cylinder and propels the piston to the end of its stroke, and the attached drill strikes the rock. At that moment the piston reverses the valve, which admits air at the lower end of the cylinder, while a ratchet and spiral device slightly turns the tool, which is being drawn back for the next blow. As the work to be done on the return-stroke is merely to lift the tool, the annular area of the piston is but half that on the other side, and little power is consumed. At the proper point in the up-stroke the valves are again reversed and the operation repeated.

The rapidity of the blow varies with the ability of the machine, and is altered to suit the hardness of the rock. The speed averages 200 blows per minute. A short stroke, light blow and rapid rate give the best progress in hard rock, and a hard blow is best in soft rock, provided the drill does not "stick" in the hole. High speed may be desirable to attain rapid penetration, but kinematic difficulties place a limit to the

speed. A maximum of effectiveness is obtained when the full air-pressure is exerted at the moment of the blow. So the valve should not reverse until that instant, and then instantly, without "dancing"; nor should there be any back-pressure on the lower side of the piston. To do this rapidly and accurately was the problem.

It is in the solution of this, the predominant feature of drill mechanism, that the Sergeant, Rand, Ingersoll, and Burleigh types have survived the active competition, in this country; the Darlington, the English favorite, accomplished it in a different manner; while on the Continent the successful native machine is the Schram.

There are two systems of moving the valves,—the tappet, requiring levers, and the duplex, requiring a fluid. The first has long retained its place. In all the early forms of drills, except the Wood, the valve was operated by means of an external rod from an exposed three-arm tappet, moved by a projection on the piston-rod. This is the principle of the steam-pump, but its slow speed does not give rise to the trouble that was found with power-drills, in which the numerous and violent shocks caused the breakage of the moving parts, particularly in a cold atmosphere. During the progress of the Hoosac Tunnel, so continual were the repairs, that a perpetual stream of men was passing, carrying some piece of the machine.

These repairs, the loss of head and of power, because the valve is reversing before the piston has completed its stroke, the danger of knocking out the cylinder-head if the tappet fails, and other early objections to the tappet, were gradually overcome. The tappets were concealed, the arc of their motion was reduced, and the form of the machine was rendered more compact. Many of these disadvantages were inseparable from the form, but the fact that a positive valve movement is obtained and that it is safer in the hands of unskilled labor, explains its retention.

In the Burleigh, the piston operates two rockers, which in turn oscillate the valve. This requires more dead-space and consumes more steam than the improvements adopted in the

"Little Giant," Fig. 234. Its valve is thrown by a centrally located three-arm rocker, that insures a positive motion. The durability was increased by separating the spindles from the

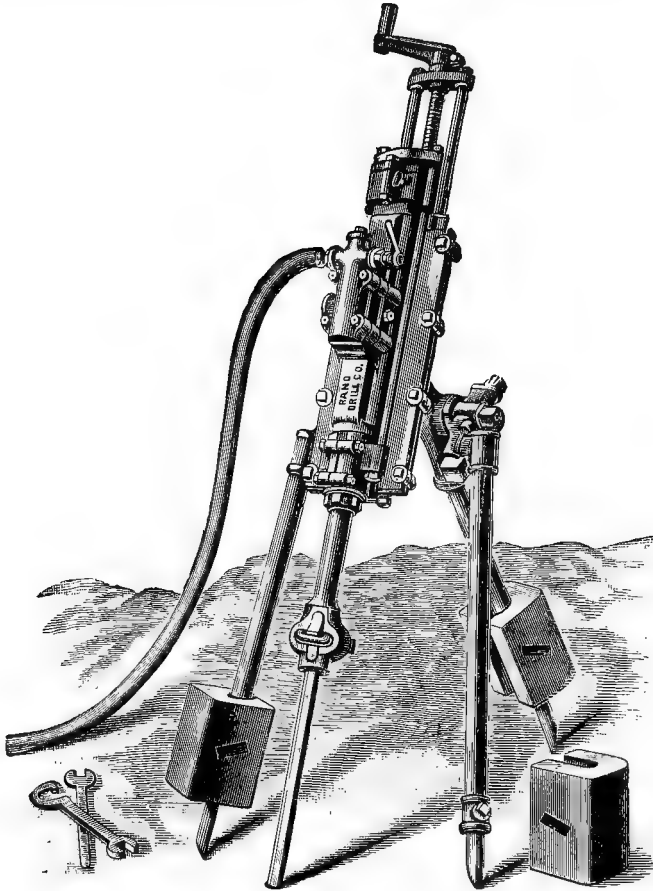


FIG. 234.

valves and tappets which they connect. The Sergeant tappet has the valve and the rocker in one three-armed piece. In both, the movement is effected by contact with the inclined planes on the piston.

The other valve mechanism is that adopted in the "Slugger," Sergeant, Ingersoll, and Schram. It embraces a steam-moving valve, which admits of higher rate of speed. The "Eclipse," Ingersoll (Fig. 236), and the Rand "Slugger" (Fig. 235), are of similar action. Two port-holes connect the annular groove in the piston with each opposite end of the valve-chest, and are opened or closed by the piston passing over them; the supply for one end, and the exhaust to the other end of the valve-chest, are simultaneously opened. The annular groove, therefore, is a general exhaust-outlet for the valve steam, while the motor steam is exhausted by the valve connecting the inlet passage with the exhaust-pipe. The sectional views (Figs. 235 to 237) show the connections clearly. When the piston and cylinder wear away slightly, the steam-pressure works to the wrong end, the exhaust becomes imperfect, and the valves fail to act properly.

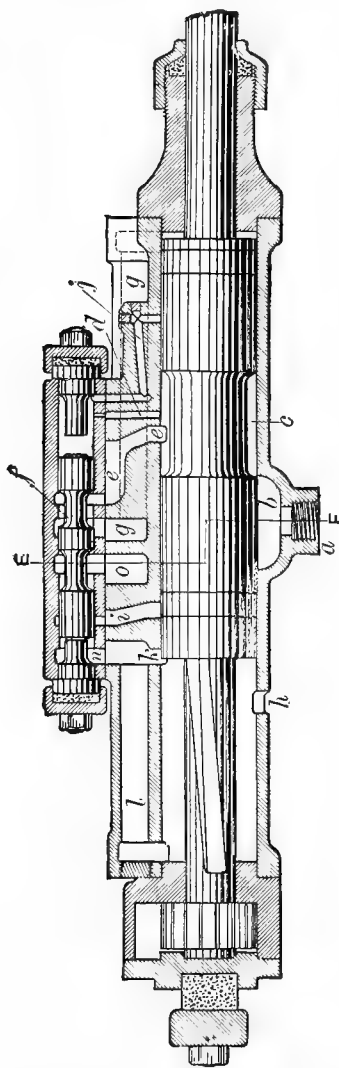
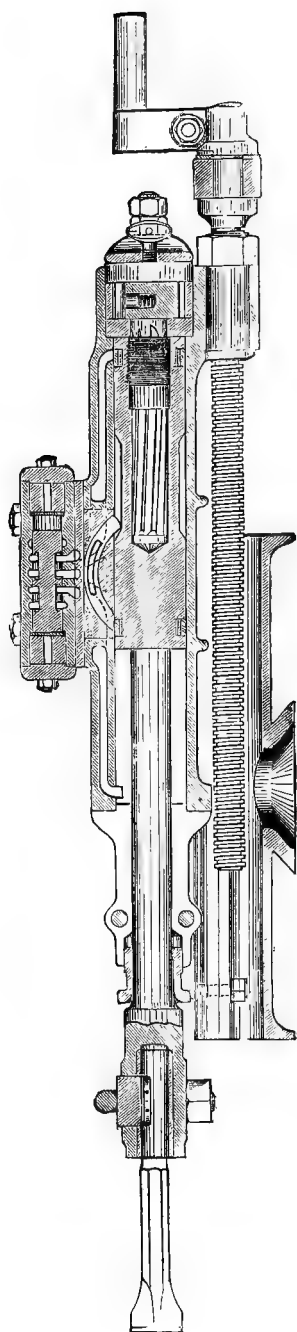
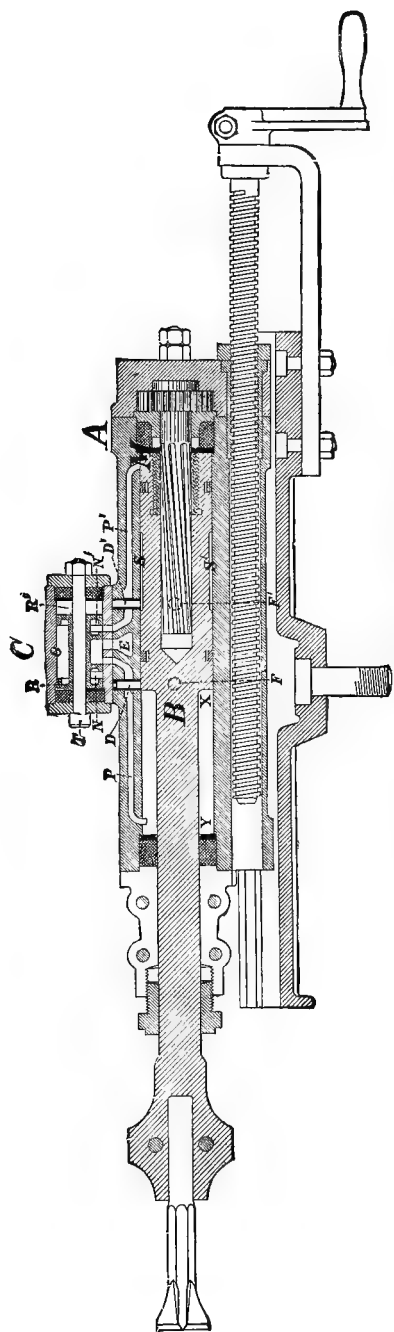


FIG. 235.

In the Sergeant (Fig. 237), the piston-valve is moved by exhaust steam from the opposite ends. An auxiliary slide-valve moves over the arc of a circle by shoulders on the piston, opens and closes the ports, and is a trigger regulating the movement of the main valve. There are no openings in



the side of the cylinder, and no ports for the piston to close; the exhaust remains open at one end till the blow is struck, when the valve reverses immediately.

An account of the Schram and the Darlington drills is to be found in "André's Mining Machinery," from which the following is taken: "Schram's consists of a slide-valve and a slide-rod that admits steam to the cylinder for raising the piston and drill. When the piston passes a certain front port-hole, steam enters through it into the back of the valve-chest, at the same time that the front valve-chest, through the other port and the hollow circular groove of the piston, communicates with the exhaust-pipe. Steam then works full pressure on the slide cylindrical rod, which, with the slide-valve, is forced towards the front valve-chest, so that the back steam-passage is open to the cylinder, and the front steam-passage connects with the exhaust pipe. The piston moves forward, and, when it passes the back port, allows the steam to enter the front valve-chest at the same time that the back valve-chest, through its back port and the circular groove of the piston, communicates with the exhaust. The slide-rod is forced back, the front steam-passage opens, and the back passage communicates with the exhaust. The slide is in the form of two spindle-valves, so that it remains in position without recoil, and the annular groove of the piston is always in communication with the exhaust.

"The Darlington has only two working parts,—an extreme of simplicity: a cylinder and its cover, and a piston and its rod. The piston is made to operate as a valve. The inlet pipe, having open connection with the cylinder, *always* furnishes the pressure to lift the drill, which rises whenever there is no pressure on the back. On its way up, the piston first covers the exhaust (above the inlet), and then uncovers an equilibrium-passage, by means of which communication is established between the front and back ends of the cylinder. Then air or steam enters and operates over the greater area, at the back, and first checks the upward movement, soon overcomes it, and finally produces a forward motion. The propelling force, now,

is dependent upon the difference of area between the back and front of the piston. On its way down it soon cuts off the equilibrium-passage and the air can only enter at the inlet; the steam operates by expansion for a short space, till the piston has passed and uncovered the exhaust-port, when a discharge takes place as the blow is being struck. One fact is noticeable, that the amount of steam used is only that necessary for the down stroke; for that used to raise the drill escapes by the equilibrium-passage to the top."

87. The drill-tool is of steel, with an X, I, Z or S cutter, the first three forms being more common, because more rapidly dressed to a shape; but they must be very regularly turned, or the hole will be "rifled" (cut triangularly instead of circularly). The S is the surest for a round hole. If there should be a tendency to rifling, try a change in the form of the bit. Each bit is of specific value. The flat cuts homogeneous rock well, but will not stand long. In sandstone, the bit should be bluff, and in some silicious rocks even have a slightly flattened edge, a "stub." For rocks that do not crush, but chip, a sharp edge will be needed. The steel used is from $\frac{3}{4}$ " to $1\frac{1}{2}$ " diameter, according to the percussion to be imparted to the rock. The smallest is for a 2", and the largest mentioned for a 5" piston, corresponding to a blow of about 200 lbs. and 1200 lbs. respectively. The drill steel is obtained in sets graded according to the amount of the feed of the drill with which they are to be used. Each bit has a life of about 275 feet of holes, of moderate depth each, provided the machine is not too powerful to handle long steel. The average mining size is the 3" piston, with a $1\frac{1}{4}$ " to 1" steel, feeding 20" to 24", and having 8 pieces to the set; the longest for a hole of about 10 feet. Ordinarily, a bit will drill 3" before requiring sharpening; and in changing the tool care should be taken that the follower has an edge narrower by $\frac{1}{8}$ " to $\frac{1}{4}$ " than the one withdrawn.

The shank of the drill steel is inserted into the enlarged end of the piston-rod and clasped by a split-chuck lock-ring (Fig. 234), or it is keyed or bolted.

The rotation of the drill through a small arc, each stroke, is

accomplished by about the same appliance in all patterns,—a fluted bar and nut constituting a ratchet. The rotation must be perfectly regular, to prevent rifling. The Burleigh has a spiral feather on the piston-rod, recessed into a groove-piece in the cylinder-head. It is toothed and held by a detent, which permits it to turn on the forward stroke, but prevents turning during the up stroke of the engine. In the Ingersoll, a grooved bar fitting into the back of the piston turns it on the back stroke, and is itself allowed to rotate on the down

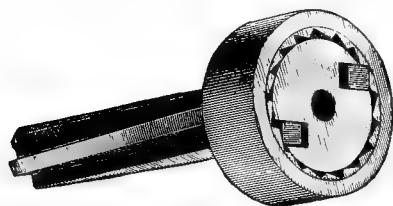


FIG. 238.

stroke (Fig. 238). The Darlington device is like the Burleigh. It turns the piston and drill on the up stroke, and itself turns during the down stroke. In the Schram, an auxiliary piston turns the drill.

In whatever the pattern, the cylinder and its tool slides in a guide-way which, being rigidly mounted, carries the drill-point forward more or less rapidly as the cutting is fast or slow. This must be done simply, and may be by hand or automatically. For mining purposes, and whenever the small sizes of drills are employed, an automatic feed is of little value, and a man is employed instead. Irregularity in the nature of the rock implies a varying rate of penetration, and, hence, a variable feed. If a fissured rock or cavity is encountered, the drill would suddenly give way, and the uncompromising regularity of the blow would result disastrously. Only a prescient feed would obviate this liability to excessive stroke. In the larger sizes of drills the piston strikes a knuckle-joint at the bottom of the cylinder and revolves a nut that feeds the drill to its work. This saves one man as each machine would otherwise require two.

The percussive effect improves as the full steam pressure is obtained at the moment of the blow ; and what is called a perfectly "dead blow" is highly desirable ; but it is inadvisable, on account of the shock to the machine and the consequent re-

pairs. The piston is, therefore, caused to terminate its stroke on elastic buffers, or against an air-cushion in the clearance-space. The latter consumes motor fluid, but is less expensive than the repairs due to a dead blow. In several forms of drills the piston is cushioned by the exhaust, instead of live air, as is the case with the plain slide-valve patterns.

A rigid support is an essential adjunct to the drill, and several types of mountings are provided, each having a special end in view, though a machine can be shifted from one style to another. In tunnels and shafts where the ranges of holes have approximately parallel directions, it is clamped to a stout hollow cylindrical column (Fig. 239), or upon a project-

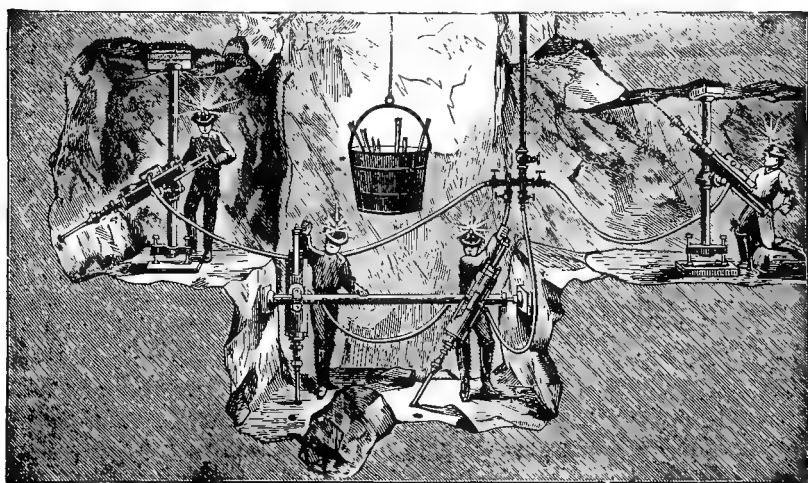


FIG. 239.

ing arm, as in Fig 226, which admits of drilling several holes from one position of support. The arm on the bar gives an eccentric range to the drill. Jack-screws at one end clamp the column, which terminates in claws that bear into blocks resting on the rock. These can be had 6, 8, or 10 feet long, weighing about 30 lbs. per foot, at \$60 to \$90.

The tripod form is the more advantageous support for surface or for stope work, where it is expected to be an acrobat

(Fig. 240). It should have a universal joint, be strong, and easily set. Each leg rests in a moiled-out hole.

This machine, as described, has no intricate mechanism to watch and manipulate, and should operate from the "go." Steam gives a little trouble in starting, because of the unequal heating of the parts, but, by proper throttling, injury is avoided. The drill should always be started on a square face. Glancing blows are ruinous. Holes should be started at short, light

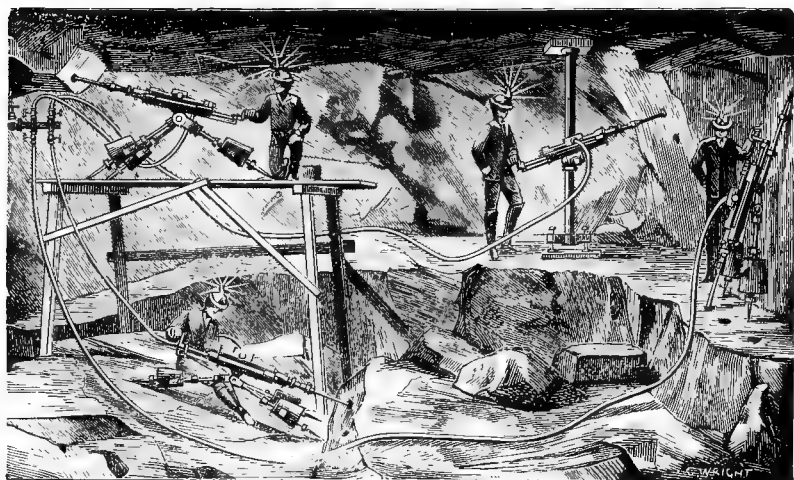


FIG. 240.

strokes; the short stroke is obtained by feeding the cylinder close toward the rock.

It is admitted on all hands now that the power-drill has passed its tentative stage, and can do more work with the consumption of less powder, steel, and smithing than can hand-work, and in any place that can accommodate a "double-hand" gang. One would not undertake to discuss the comparative excellences of the different drills on the market. There are several styles, doing all manner of work at shafting, tunnelling, and stoping; Figs. 239 and 240 illustrate the manner of their use. Personal observation among, and discussion with, operators in various districts fail to reveal any formula by which the makes may be gauged. In one camp the Rand, in

another the Rand, Waring, and National, are indiscriminately used; still another prefers the Burleigh; while in others out here the Ingersoll excludes all others. One region prefers the "Little Giant," and another mine will discard it for the "Slugger;" in like manner preferences are displayed for the "Eclipse" or the "Sergeant." They are all highly commended, and their employment in a particular locality may be a matter of accident or of natural selection, the rock happening to be most suitable to the given form which has then survived the periods of test.

Certain it is that the author's experience favors the fluid-moved valve-drill for hard rock, the Slugger and Sergeant being adapted to our Rocky Mountain material; but whether or not they are under all circumstances the best, one would not dare to aver. Each miner must determine, from the nature of his rock, the proper air-pressure, rate of speed, and proportion of rotary motion required for the most effect. The manufacturers can give great assistance in this regard.

The comparative tests announced by different makers are of too short a duration, and are conducted under conditions too limited to avail the engineer. As a matter of fact, it becomes a question of the survival of the fittest, and that is determined by the success with which the essential attributes are supplied. A stated air-pressure will accomplish a certain penetration in an ideal drill, but the various patterns will approach this amount more or less satisfactorily as the frictional resistances are less, if the blow is uncushioned, and if the reversing-valve is perfectly accurate.

Of course, the heavier the impact, the greater the effect; but the blow is dependent upon the pressure and the drill weight. For hard rock, therefore, either the pressure should be high or the moving mass large. The former is inexpedient for economical reasons explained in No. 93, so a heavy striking mass is imperative.

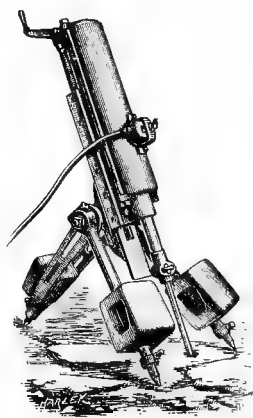


FIG. 241.

On the other hand, power-drills should be portable, necessitating a light frame and guide. A high piston-speed may be desirable and advantageous, but the kinematic difficulties render it unadvisable.

Besides these qualities, however, are those which never figure in the comparative tests, so called, the convenience in handling, and true automatic rotary and feed appliances. If a machine is capable of a variable stroke, so as to start the hole on a light, short stroke, and will "mud" well, it meets two very important features that are not always possessed. A long stroke conduces to quick mudding.

In remote camps the dominant attribute is a simplicity of parts to assure a "lasting-capacity" as well as a "boring-capacity." The early pattern is said to have had 80 pieces in it, and its repairs were so numerous that each drill was built over every two years, and it required five machines to keep one going. Now continuous work is maintained with one drill in the shop while two are working, its average life being 8 shifts, corresponding to about 400 lineal feet of holes. In a certain Lake Superior copper-mine the cost of blacksmithing is about 64 cents per drill per 24 hours. The amount and cost of breakages are too variable for any precise estimate. A mine employing 22 drills constantly allows for \$60 annual repairs per drill. These two items, amounting daily to 85 cents per machine, may seem an unfavorable comparison with hand labor, where 55 cents was the allowance per daily gang; but a reduction to the relative progress will prove more equitable.

A recital of a few of the comparative tests may be of interest. About Silverton, where 7 inches of hole will dull 14 to 20 drills, a machine cut 2 feet, the length of its lead, in 12 minutes. Three men will drill three 30-inch holes in 10 hours, while a machine does seven holes of $5\frac{1}{2}$ feet each. An average of nine neighboring mines, in the conglomerate, showed machine drifting and stoping to be, respectively, 22 and 36 per cent cheaper than hand, and sinking 4 per cent dearer, with a progress 60, 54, and 38 per cent more rapid, the latter gain in sinking compensating for its increased expense. In the iron-mines, machine labor is one fourth as expensive as manual. Three

men on a 6×16 shaft did 0.37 feet daily, while two machines advanced 3.4. H. S. Drinker, "Explosive Compounds," quotes an average daily progress by hand and black powder, in 21 tunnels driven in solid hard rock, of 1.441' in heading, and 1.96' in the bench; and of 58 tunnels in easier rock, 2.55' and 2.62', respectively. With machines and nitro-glycerine the progress was five to seven times as fast. The recently completed Cascade Tunnel made 2 lineal feet per 24 hours with 17 men on the heading, and 6.9 feet with 5 machines.

An eleven-months comparison of hand, Schram percussion-drill, and Brandt's rotary drill, gives an efficiency as to speed of 1:4.73: and 5.26, relatively; and as to cost, 1:0.62: and 0.60.

The consumption of fuel and air per drill may be calculated as in any ordinary steam-engine. The cost of the ordinary mining-drill is about \$325, and of a complete plant of 6 drills, with a 16×24 compressor, etc., is \$7000. A smaller outfit for 3 drills was recently delivered in Denver for \$3700.

There are several patterns of percussion-drills operated by electricity, but the results give as yet insufficient proof of its value for reciprocating machinery. In Fig. 241 is shown the Edison drill.

The average depth of holes in tunnels rarely exceeds twelve feet; in stopes and narrow work, four feet. A very deep penetration cannot be obtained; the impact of the blow would be destructive to a long line of rods, and the drawback power of the piston is small.

88. M. Leschot has the credit of the first application to the miner's art of rotary diamond-drills, which have since steadily gained in favor and increased in range of utility. Several diamonds are forced into sockets on the end of a steel tube, and on a rapid rotation abrade the rock. The cutter-face is entirely covered with diamonds in such manner that no concentric circle fails to touch one, and one or more projects transversely beyond the tube. The bit may be annular (Fig. 242) or solid convex or concave face (Figs. 241 and 244). The first is more commonly used, as by that means a central core of rock is uncut, and may subsequently be withdrawn for inspection. The débris is carried away by means

of a stream of water passing down inside of the tubes, washing the drill-face and carrying the cuttings up outside. The solid-head bits are preferred for mere drilling, except for large holes,



FIG. 242.

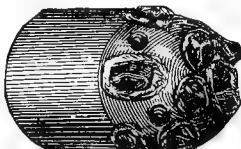


FIG. 243.

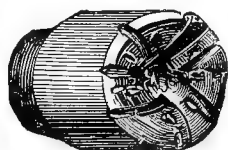


FIG. 244.

the concave surface being better than the convex. The wash water escapes through the holes in the face.

The diamonds used on the face are of the black or deep red variety; on the outer edges, borts (imperfect diamonds). "Theoretically, too many carbons cannot be put in; there should be never less than 12," and as many as 20 may be mounted on a bit. Recesses are accurately prepared for them, into which they are set and secured by metal hammered up around them. In some cases a firm setting is obtained by forcing the stones forward through small holes in the metal by means of a screw, or by hydraulic pressure. A later method consists in forcing the stones nearly through the metal, and subsequently grinding the steel down until the stones are exposed. The bit is coupled to the tube, which is added in 8-foot lengths as the hole deepens. The diameter of the hole is a matter of indifference where prospecting or the long-hole drilling is intended.

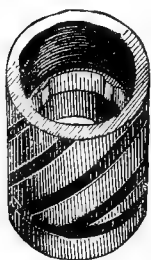


FIG. 245.

Those of ordinary depth are up to 3 inches diameter, and those of great depth taper from 5 inches down. The tubes are of slightly smaller diameter, $1\frac{5}{8}$ inch tube is used in $1\frac{3}{4}$ hole, and weighs 3.4 lbs. per foot. Figs. 245 and 247 show the guide, which is just the size of the hole, and maintains the bit in the direction in which it started; the spiral grooves allow the water to escape.

At the upper end of the drill rod is a joint or swivel, through which the supply of water is forced by means

of a pump. Above is the connection with a rotary and feed motor operated by a steam-engine, the capacity of which varies with the amount and size of drill-tube to be manipulated. An 8-horse-power engine is suitable for a 1000-foot bore-hole. The running-gear should be firmly framed and supported, that the weight of a great line of rods may be easily handled; 1000 feet will weigh from 4500 to 6000 lbs. A very light temporary shed will suffice for cover.

Two methods avoiding a positive feed are in vogue for driving: one, a spur-wheel feed; the other, the hydraulic. The former is so adjusted by differential gear that its friction shall equal a desired resistance; and when this is exceeded, because of undue strain below, a regulation is obtained. Stratified rock changes so much and so rapidly in structure that a uniform feed is impracticable in deep holes, and inferior to the hydraulic feed, of which Fig. 246 is a section. It is a simple motor, which by means of hydraulic pressure on the piston produces a pressure which is maintained constant. Both ends of the cylinder are connected with the pump, and suitable cocks admit of a perfect control by the operator, who gives any variation or reversal of speed within the limit of the pump and piston-area. Gauges indicate the

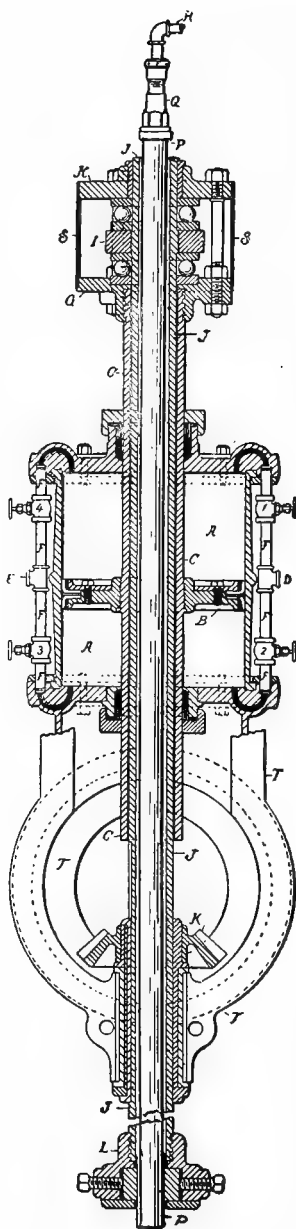


FIG. 246.

pressure. Only the hardness of the rock determines the rate of feed, and this rational system saves all parts of the machine from danger of breakage. Fig. 247 shows the feed-cylinder as the extension of the drill-tubes. The connection between the tube and the feed is by some form of chuck, which may be

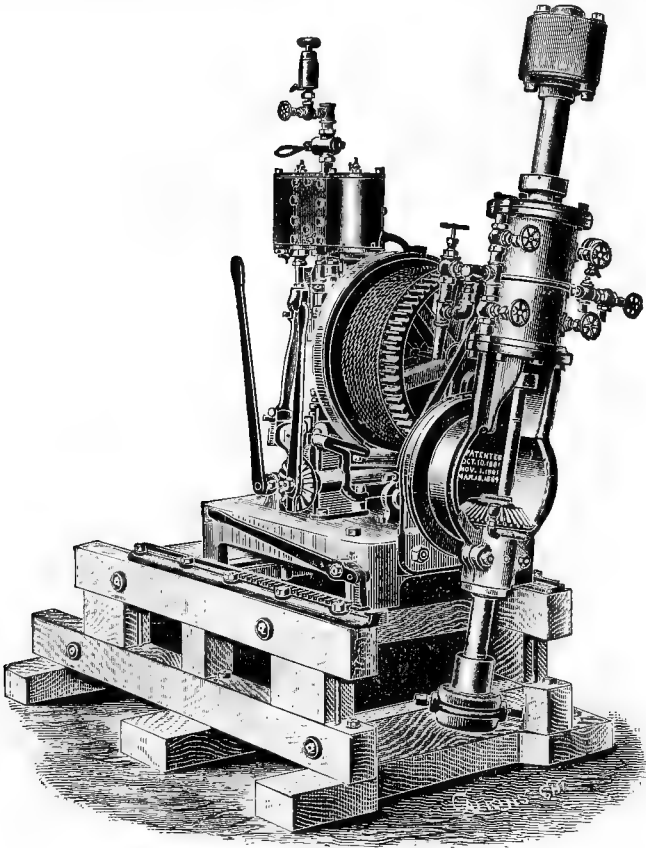


FIG 247.

loosened at the end of the feed-stroke and run up to the top for a new grip.

The pressure exerted by the feed is just sufficient to produce abrasion, not to cut the rock. The tube is partially suspended by friction-rollers at the surface, so that it is subjected

to very little tension. The power producing rotation must be less than the torsional strength of the rods. This would place a limit on the possible depth of explorations, while the regulating power of the feed limits the capacity of the machine.

In addition to the integral parts mentioned, a steam-engine, gear, and hoisting-drum are compacted upon a rigid mounting, varied with the purpose of the borer. It may be bolted to a heavy frame bed placed on wheels, with portable boiler, or mounted as in Fig. 247 for underground work. One form is



FIG. 248.

of gun-metal and steel, and weighs only 400 lbs., yet can bore 150 feet with ease. The drum is added for hoisting the drill-tube without altering the position of the machine, which remains in place till the bore is completed. A high derrick facilitates the addition or disjointing of tubes.

The rate of revolution of the tube and its bit is from 400 to 800 per minute, and the progress is remarkably fast, averaging a penetration of 13 inches to 2 feet per hour, stops inclusive. The drill bores only about one half the time. The use of the annular bit does not increase the speed, for the rods must be raised every 10 to 15 feet of advance to examine the core, which is broken from its place by the core-lifter (Fig. 249), and raised with the tube. In uniform rock the tool need not be raised as frequently as in strata of varying texture. Should the hole have penetrated a soft layer between two hard ones, the core would twist off and grind it away, and its existence would not be made known in the core. Again, the tendency of the core to turn in its tube would give false information as to the dip of the strata. For this reason, also, a flat, not round, hoisting-rope should be used. At best the core is only a partial guide. A slime-box receiving the cuttings would indicate the presence of the soft rock, but many causes combine to make even this examination unreliable. A careful measurement, an



FIG. 249

allowance for wear, and frequent raisings are the only checks. Shales and clay slates give smooth sailing, but fire-clay chokes the barrel. In such cases the full pressure of the pump will usually wash it out ; if not, the tube must be lifted. With holes of a moderate diameter there is no necessity for tubing the hole.

Accidents are rare. A diamond may fall out, and, if it can-

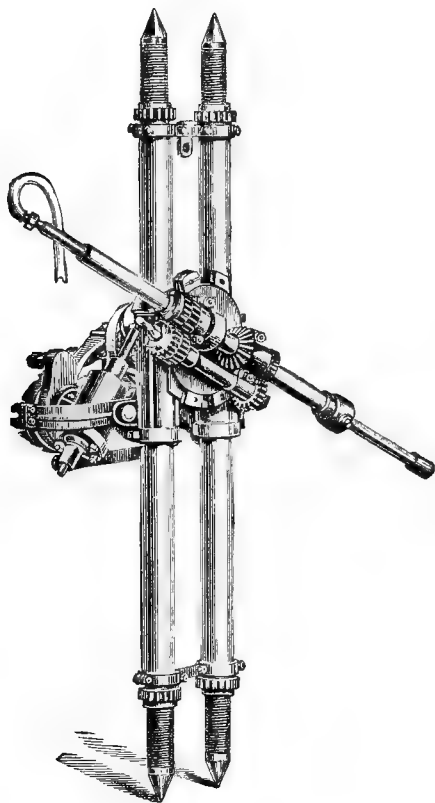


FIG. 250.

not be recovered, must be chopped up at once, or the water supply must be reversed to wash the stone up the tube. A chopping-bit is used to break up hard nodules or boulders.

89. Holes may be bored in any direction, though the machine is best adapted to vertical ones. Fig. 247 shows the

machine drilling at an angle; the "Little Beauty" (Fig. 250) drills 70 feet horizontally without trouble. The friction of the tube on the rock limits the length of flat hole that may be drilled.

In Fig. 251 are exhibited explorations in the Silver Isle

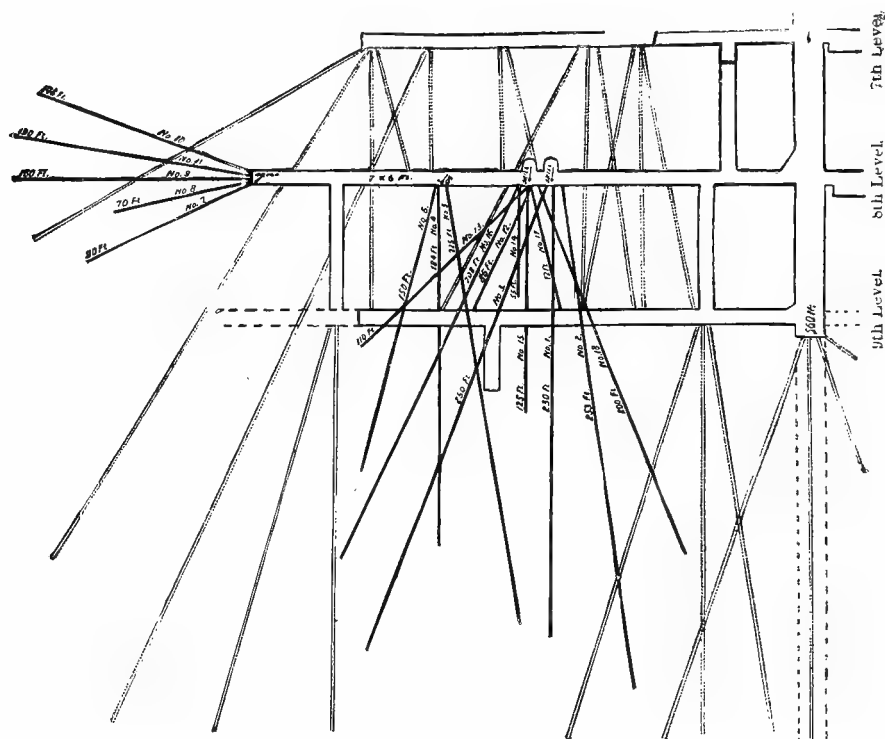


FIG 251

mine by the use of an underground machine. For prospecting territory, for drilling a deep sump-hole to drain a mine, for rapidly sinking a connection through which to pump out a drowned mine, to sink a tail-rope bore-hole, etc., the utility of the diamond drill is generally recognized. The Poetsch method (p. 253) depends upon it, and the long-hole process is possible only by the use of it. It is suitable in hard or the hardest

rocks, and, remarkably enough, will perform in granite better than in soft stone, according to the report of the Superintendent of the Hope Mining Co. Doubtless many properties owe their existence to the result of diamond-drill discoveries, and its use has frequently saved expense in various ways. But it is not considered infallible in its indications as to the presence or absence of the ore body sought. Though it is true that tunnels have been carried by the long-hole process at home and abroad, the percussion-drill is cheaper in tunnel and for short holes. The cost of drilling varies materially. An average of 29 2-inch holes, 400 feet each, was \$2.35 per foot in a Lake Superior iron-mine; 16 holes, aggregating 5877 feet, cost \$1.97 in the Pennsylvania coal measures; and 24 holes, averaging 18.9 feet per shift, with a total of 9902 feet, cost \$2.22 per foot. In the Mariposa estate, the cost of prospecting holes 74 to 231 feet deep, in 34 to 146 hours, averaged \$1.10 per foot, including diamonds (\$0.32). The actual drilling time was about one half the total.

There is a great difference in the item charged to wear and tear of the diamonds, varying from 21 cents to 56 cents per foot. Experience has determined that the diamond is practically useless after 6 settings. Manufacturers say that there is a remarkable difference in the quality, hence in the wear of the stones. The borts and black stones are tougher than the vitreous. The item does not refer so much to the wear of the stones—as that has been found to be inappreciable after drilling 400 feet, but rather to the loss due to the falling of the stones out of their sockets. Ground charged with pyrites is especially bad, causing the stones to crumble.

Comparing it with other methods, the diamond drill is rarely cheaper in deep soft rock than the Mather and Platt system (p. 397); in hard rock it supersedes all others, except where water is very scarce. Tubing in conjunction with it is troublesome, if not out of question, for deep holes, and reaming is not easily done.

Cost of drill and outfit for 1000 feet of 2" rods, \$3872. Two drills require 5 men.

For underground work a 3-horse-power electric motor is mounted on a truck, with drum, drill, and pump, and permits core-drilling to advantage in small spaces. In many mines 1" cores in sections of 5" to 20" are cut for 80 feet depth, and a great deal of prospecting has been prosecuted with this compact machine, which makes 1.60 feet per hour at a cost of 68 cents to \$1.03 a foot. Fred. G. Bulkley, of Aspen, Colo., has devised a graphic representation of the results of borings by plotting them to scale on a cross-section paper, which pictorially conveys the information as to seams, faults, etc.

Rotary perforators for tunneling-out the full area of headings and entries are offered on the market. At one operation a series of cutters on a rotating boring-head grinds away the whole face for a core from the heading some 7 feet in diameter. One was used in the Mersey subaqueous tunnel. It travelled at the rate of 39" per hour, and executed its work satisfactorily in the argillaceous chalk.

Brandt's borer, which is highly esteemed in Prussia, is a hollow cylindrical steel bar, on the end of which are formed five teeth. Rotated by a pair of small hydraulic engines, it is forced against the face of the rock, and cuts a hole the core of which is cleared away by the continuous stream of water escaping from the driving-cylinders.

90. Since the advances made in the manufacture and use of the machine-drill, the systems of drilling and of blasting have had to undergo corresponding changes. In hand-work, the object sought is as much to secure a good bench for the next shot as to break ground with the present. With simultaneous shooting, and particularly in tight ground (on faces of drifts or shafts), all of the holes are drilled more or less axially, and the blasting operations are conducted differently, because the inconvenience of handling machines supersedes the gain from attention to the lines of least resistance, and it is not always possible to drill holes with the machine in such a way as to conform to the fundamental principles.

According to the mode of arranging holes, we have three systems. The first was employed with the earliest experi-

mental work on the pioneer machines at Mont Cenis and St. Gothard tunnels. Eight perforators were mounted on a car-

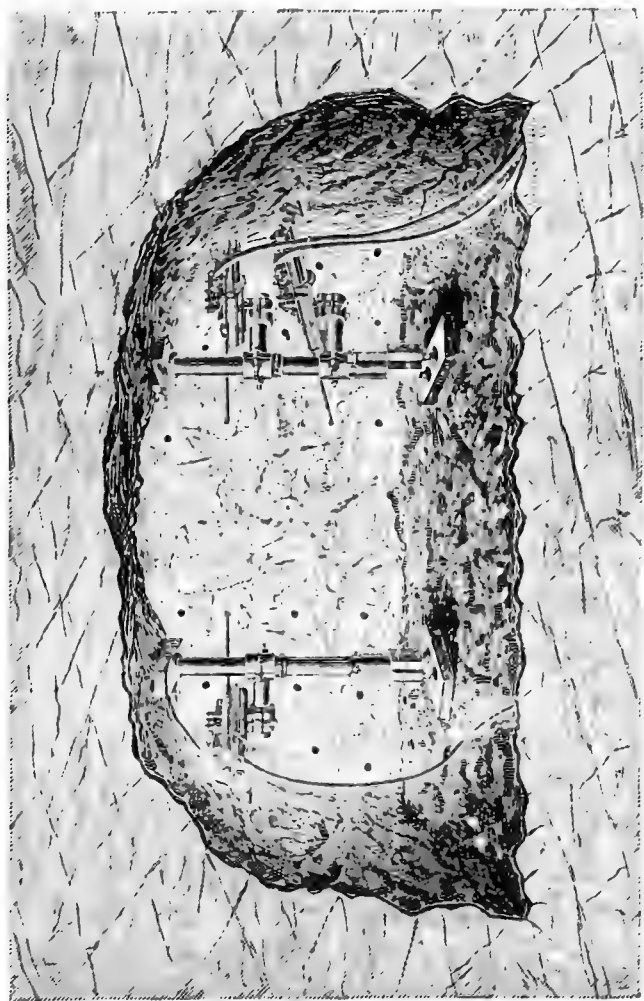


FIG. 252.

riage, and bored holes at different angles covering an area of 250 square feet. When the requisite number of holes was drilled, the machine was shifted to another space, where it

repeated the performance. It was run away when the firing was to be done. A centre hole was surrounded by a ring of eight rupturing-holes, outside of which were 3 full and 2 segmental concentric rings of holes. These were fired in volleys after the first central set. With 18 holes of 3 feet to 5 feet each, charged with $1\frac{3}{4}$ lbs. dynamite, the progress averaged 18 feet a day through schists and gneiss.

The second system is very popular, and known as the 'centre-cut,' which was introduced in the Musconetcong Tunnel, increasing the progress from 89 feet to 116 feet per month. The American method of tunnelling was in process (see Figs. 201 and 254, and p. 388). The face of the heading was 8 feet high by 26 feet wide, and had six machines operating on it, drilling 36 holes of $1\frac{1}{2}$ " to $2\frac{3}{4}$ " diameter. The holes are drilled in vertical rows of four each, and a depth according to the location. The two central rows "look toward" each other, and meet at the bottom (Fig. 252). The next two rows on each side of the axis also point inward, but less so than the central or cutting rows; while the outside rows are parallel to the axis, or incline slightly outward. Roof holes and corner squaring-up holes complete the drilling, and should trim up the profile of the tunnel at once. The positions of these holes are variable. In very hard rock the holes of the two central rows are in pairs close together; sometimes they are single, but large, 4" diameter. In firing the two central rows (1, 1, Fig. 253), first break out an entering wedge,—not to the bottom of the holes,—which facilitates the work of the next two rows (2, 2), which shoot toward the walls, after which the advance is squared up. The breaking-in is done with electricity, but the enlargement and squaring-up is done by fuse and a lower grade of explosive.

The depth of the holes and their distance apart depend upon the rock and the advance desired. Advances of 14 feet have been made, but there is a limit to the capacity even of nitro-glycerine, and 10 feet is quite sufficient. To secure this, clean, the two central rows of holes are $10\frac{1}{4}$ feet deep, the remainder 12 feet, except the six roof-holes of 8 feet each. In a narrower

tunnel of say 11 to 16 feet wide, an 8-foot advance will suffice. Four machines can easily operate in a double-track (27 feet) tunnel—six can be arranged by placing two on each of the two central columns. Three machines in a single-track tunnel, and two in an 11-foot heading, will give progress as rapid as the shovellers can handle the dirt. Out of an average 8-hour shift the actual drilling heat is about $5\frac{1}{2}$ hours; the shifting of tools,

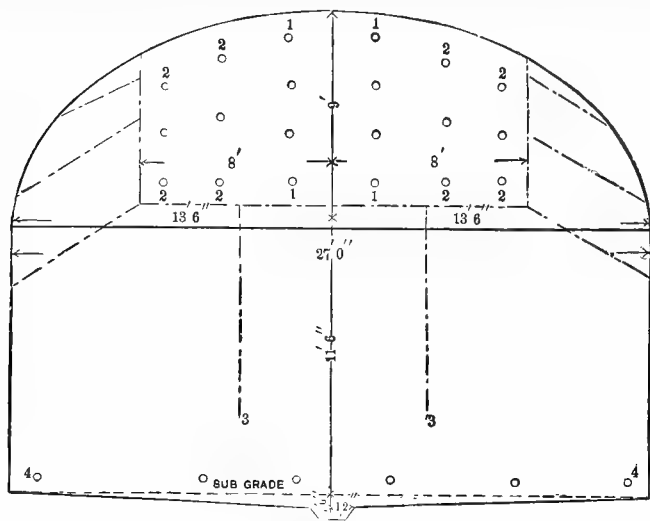


FIG. 253

etc., takes $\frac{1}{2}$ hour; loading and blasting and removing rock, about an hour each.

As illustrations of progress we have: the heading of the Haverstraw Tunnel, 9×16 , requiring 20 holes of about 8.4 feet each, was completed in 20 hours; weekly progress in 8×27 South Penn Tunnel, 74 feet of sandstone; two machines in Washington Tunnel, $7\frac{1}{2} \times 11$, progressed 8.26 feet per day in solid rock, with 26 holes of about 10 feet; the Cascade Tunnel, 16 feet wide by 22 feet high, progress 200 feet per month, with two faces of attack, 20 to 23 holes 12 feet deep, by 5 machines; in medium hard basaltic rock, average 6.9 feet per 24 hours; four machines in the Vosburg heading, 8×27 , made the advance

in 10 hours, with 26 holes (8 centre and 18 sides); in the D. & R. G. R. R. Tunnel, two machines made a complete drilling round of 20 holes, 9 feet deep, in 7 hours; the aggregate depth of the 36 holes in the Musconetcong Tunnel was 408 lineal feet, the firing of which gave nearly 10 feet advance; one shift drilled and broke a cut or a side round with six machines.

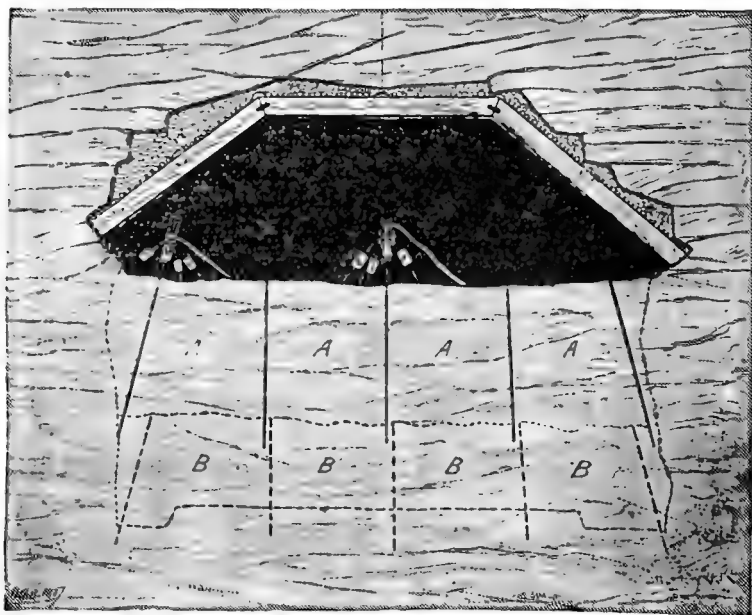


FIG. 254.

The consumption of powder varies. 7 lbs. of Giant No. 2 was used in the centre cut of the Washington Tunnel, $5\frac{1}{2}$ for each side round hole; the Musconetcong consumed 0.4 lb. of nitro-glycerine, and 4 lbs. of Giant No. 2 per cu. yd. broken; on the Mariposa estate, in very tough rock, 7 lbs. of Hercules No. 1 and 10 lbs. of No. 2 per lineal foot of drift; in the Vosburg, 100 to 120 lbs. of Rackarock per advance.

The bench of the tunnels is attached in one (Fig. 253) or two (Fig. 254) sections, *A* and *B*; two wall holes, one or two

transverse rows of 4 top holes downward, and half a dozen bottom holes, lift each bench with every other shift. Fifty-four feet a week is the record on a very hard sandstone bench, 14×27 . This work is not only more rapidly accomplished, but also with a powder consumption per cu. yd. of rock of about one half that in the heading.

91. Brain's radial system is employed in headings too small for more than one machine, and, like the "centre cut," is equally applicable to shafts. The design is to drill all the holes from one position of the machine, and thus minimize the time lost in shifting. The holes are shallow and vary greatly in length, those making the smallest angle with the face being the longest. Four ranges of holes are drilled, and in a certain case the machine, from a position $4' 8''$ from the bottom, $2'$ from the top, and $2' 6''$ back from the face, put 29 holes with a total length 70', advancing $3'$ with an average of 2.4 cu. ft. broken rock per lineal foot of hole. Sometimes a few extra squaring-up and lifting-holes are necessary to trim the periphery of the drift, but, ordinarily, the firing of the most angling holes first breaks out the rock to daylight and opens a face for the other successive rounds. The advance cannot be large, for neither deep nor angling holes are possible in a narrow drift. In a drift $8'$ wide, two settings of the machine are sometimes made drilling from near each wall, and thus forming a modified centre-cut plan. In some mines a practice prevails of cutting a horizontal range of bottom holes, two ranges of holes looking downward, and a top row to break out horizontal instead of vertical wedges; this plan requires a bar-mounting for the drill, and a drift say 7×8 feet.

Gen. Henry Pleasant's method of shaft-sinking is a novel and eminently successful application of the diamond drill. One or more diamond drilling-machines are set up over the site of the shaft, and bore vertical holes as deep as the shaft is to be carried. The machines are moved to new positions and additional long holes bored. The operation is continued until the entire area of the shaft is pierced by holes at suitable distances apart. The St. Clair shaft of the Reading Coal Co.

had 35 holes drilled to a depth of 200 feet; 25 holes covered the space $13' 10'' \times 16'$ of the Norwegian Colliery shaft. An average of three machines in six weeks bored 35 holes through 300 feet of hard rock over an area of $25' 8'' \times 13' 10''$.

When the "continuous process" is completed, the machines are removed for the blasting. The holes are filled with sand or water for the full length, except in the upper 3 or 4 feet, which are treated like short holes, charged with dualin and fired,—the central ones first. When the débris has been cleared away, the shaft will have advanced 3 or 4 feet. A few feet more of each hole are cleaned out (sometimes the bottom plugged with clay), loaded and fired. Thus each section advances with an alternation of shooting and hoisting. Herein lies the secret of the success of the method. The operation of boring is continuous to the end, and the other operations may be uninterruptedly prosecuted. Though it is not always cheaper per cubic feet, it effects a great saving in time, and quick access underground may prove the element essential to the success of the undertaking.

92. The undermining of coal is accomplished by machinery of two types, one dependent upon abrasion produced by a saw and chisel cutter, the other upon percussion. Of the first variety there are three general classes of machines using either a rotary bar, a rotary wheel, or a chain. The motor power for any of these types is electricity or air. The competitions in the coal trade have prompted many an engineer to turn his attention to the direction of an appliance which might, in some degree, tend to decrease the cost of production without in any way increasing the risk to life or property. These machines, of whatever type, it must be admitted, do the work of undercutting the coal to a moderate depth in a narrow groove as expeditiously as can be done by manual labor, removing therefrom the severer forms of the miner's toil, and rendering his occupation less laborious. The manual effort of the digger, exerted under the unfavorable conditions in a constrained attitude, is most wastefully applied, with the production of an excessive amount of waste

dust and small coal, and thus affects seriously the output of the mine.

The machine which is to replace manual labor must occupy little room, be low and light, capable of being handled by two men, and of a size small enough to admit of working around and between the props. It should be capable of starting in the corner of a pillar or loose end, and of cutting clean to the walls of the room, right handed or left handed, and to any height. It should be equally suitable for a vertical shearing of the coal, as well as for holing. During the year 1896, over fourteen per cent of the total bituminous coal tonnage was mined by machines in eighteen states of the Union. The U. S. Geological Survey announced the use of 1139 machines in 115 mines in fifteen states; but with the addition of other known cases, the aggregate number in use may be safely placed at 1150 machines in 120 mines producing 12,000,000 tons of coal. During 1891, the machine output was but 3.27 per cent of the total.

The machine does not dispense with the labor of the miner: it only more efficiently accomplishes the most arduous part of his work. The chief value of the change lies in the subdivision of the labor formerly imposed upon one man, and the consequent celerity and safety resulting from the attention of each man to his own branch of the work.

In Illinois the Legg machine is used in driving the rooms; elsewhere, the Harrison, Jeffrey, Yock, Lechner, and Sergeant. The Lincke is used to some extent in the western country; the Marshall and Frith is the old-style machine still in vogue in Europe.

The rotary or chisel-cutters are always accompanied by a positive feed which advances the machine; in this class are the Lechner, Jeffrey, Marshall, Hurd and Simpson, Baird and Lincke. In the percussion class are included the Frith, Harrison, and Sergeant, each of which, except the Frith, must be moved after the bearing under hole has been drilled. The Harrison machine is the most popular in Ohio and Illinois, and is illustrated in the accompanying cut (Fig. 255), which requires no explanation.

The valve-motor is a single-cam rotary device. It is compact, light, and will "bear in" about 80 lineal feet of $3\frac{1}{2}$ -foot holes in ten hours, allowing two hours lost in changing bits and positions. In twenty minutes it will cut along the face to the width of its board. 10 cu. ft. of 70 lbs.-air, at a rate of 200 blows per minute, is the average consumption. The several sizes of machines differ only in power and depth of groove.

The kit of tools (three pairs of augers 2', 4', and 6' long, and one pair of 18'' extenders) is dulled every day, and refaced, by the blacksmith. Each bit is refled by the blaster after use. With two machine-men, it employs five loaders and a blaster. The Sergeant rock-drill has been adapted also to coal-mining, and gives eminent satisfaction in the South.

Frith's machine imitates the miner, working a 75-lb. pick by bell-crank lever. At a rate of 70 blows per minute, 11 square yards of a 2'' groove are cut 42'' deep per hour. The simplicity of these patterns enables them to be readily handled in thin seams.

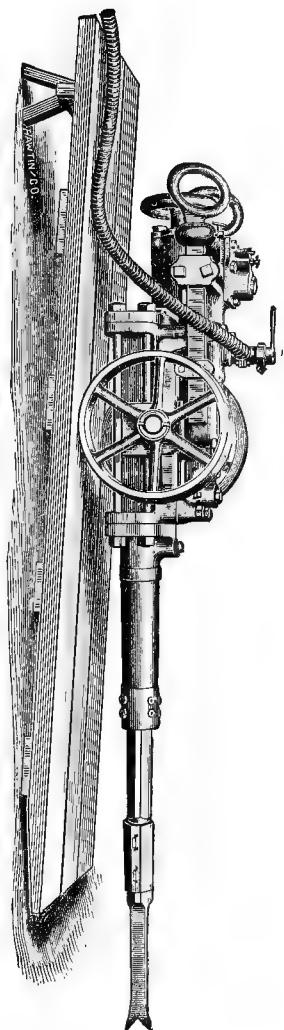
The other class of machine is represented by the Jeffrey air and electric, of which the latter style is shown (Fig. 256).

In 6 minutes it will cut a groove 39'' wide to full depth; can be reset to position in 9 minutes, and moved into an adjoining room in 20 or 30. It therefore undercuts a room in about 2 h. 10 min.; 60 amperes at 250 volts will operate it. It occupies an area of $2' \times 7' 6''$ and weighs a ton.

The Lincke cutter is a revolving axle 3' long, like the Jeffrey, and gives nearly equal satisfaction. The Lechner is similar.

The Winstanley is a rotary toothed disc capable of being turned under the carriage or out against the face, revolved by two oscillating cylinders working at a pressure of 30 lbs., and cuts 70 square feet per hour. It is mounted on a carriage moving along a track longitudinal with the coal-face; and weighs 1500 lbs.

A chain carrying several chisel cutters is the device in Marshall and Carret's machine on wheels. It is braced to the roof, and the cutters are so set that they carry the scrapings out.



Outside view.

FIG. 255.

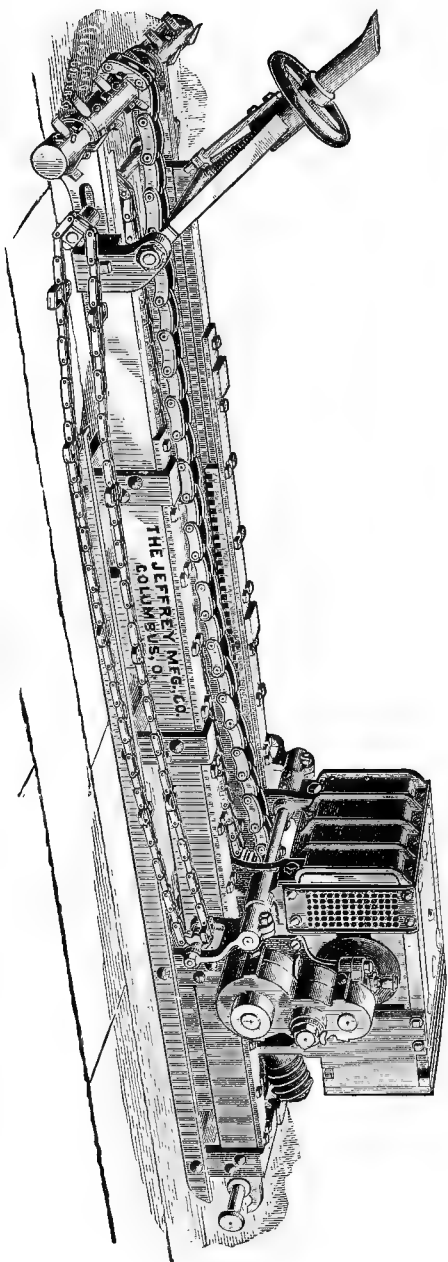


FIG. 256.

wards. This has the advantage of keeping the machine to the coal. Another Marshall design, also hydraulic, "cuts into coal like a scoop into cheese."

The Jeffrey endless chain or belt carrying a number of cutting knives and travelling horizontally about the frame of the machine is very much used, and has largely supplemented the rotary bar type. The action of the bar is somewhat similar to the cross-cut saw, because it cuts the coal across the grain, and not with it. The machine with the bar in front, Fig. 229, can be used conveniently only in headings, as it requires so much free space between the coal-breast and the timbers, for use in mines working by the long-wall method, the bar of the machine projects at right angles from the frame and because it is in closer contact with its work exerts a greater force than does the rotary-wheel which is frequently used. Both classes of bar machines have a great disadvantage in the tendency "to climb in the coal." The rotary-wheel machines and the chain-cutters act more like rip-saws, cutting along the grain, and not across it. The rotary-wheel machine, however, has a disadvantage during transportation, in occupying so great a space. If the wheel could be made in halves, fastened together by bolts easily removed, the machine would be far more acceptable in small rooms. The machines, when properly adapted, appear to work with equal facility in rooms or on long-wall in thick or thin seams, but the pillars are not yet as economically robbed as by hand. Coal containing much pyrites and bony nodules gives special difficulty to machines of the rotary bar type, but is not so great an annoyance to machines which distribute the wear and tear over all the cutting parts equally, as in the case of the rotary-wheel or the chain.

Chief Inspector of Mines for Ohio, R. M. Haseltine, reports the result of a series of investigations upon electric coal-cutting machines in the bituminous mines of Ohio, in the course of which he states that few coal-seams are adapted to machine-mining at all; and in a still smaller number can the present type of standard machine be used with economy.

For profitable mining the roof must be strong and free from slips or bell-shaped balls; especially is this true when the props are set more than twelve feet from the face to accommodate the electric cutters. The floor of the coal face must be nearly, if not quite, level to admit of the successful work of machines. The thickness of the face has been considered as the index by which its adaptability for introduction is to be determined; and, with but few exceptions, no attempt at an installation has been made except in the very thickest of faces, for which machines builders have designed. The weight of the machine for the thin vein must be as great as for a thick one, if the coal fibre in each is equally firm.

In measuring the horse-power which was necessary to drive these machines, an electric current was opened near the machine, and meters registering potential and current were inserted, as a result it was found that the bar machine consumed from 16.5 to 22.7 horse-power, while chain machines required from 8.6 to 18.1 horse-power. If the number of cuts is taken into consideration, it is found that the average horse-power required by the bar machine is 18.7, by the chain machine 14.4. The chain machine was found also to consume less power for its own driving. The relative efficiencies of the several machines in horse-power required to undercut one square foot of coal in one minute, was, for a chain 4.2, and in the bar machine, which makes a misapplication of its power, 8.9.

The introduction of a properly selected cutter always results favorably, though some mines have abandoned them for causes not pertinent to their economy. The men do not take kindly to them; but their efficiency is undoubted.

Though the number of mines using coal-cutting machines exclusively in Illinois has decreased slightly between the years 1888 and 1895, the number of machines remains about the same, and the percentage of total coal output the same. A few additional mines have introduced machines during 1895. Naturally its most general application will be in hard mining coal, which cannot be blasted "off the solid" as is soft coal.

Many of these machines can shear the coal as well as undercut. As all entry and narrow work requires shearing before breaking down, it is evident that the scrapers and planers have a decided advantage over the percussion-machines; but even they are not as economical as hand-drills. The latter progress faster, and prepare the face for the other operations quicker than do power-drills. The hand-working miner can shift his place of working from one narrow place or bench to another more readily than can the machine-cutter.

Any machine will cut 13 square feet in, say, 10 minutes, or a 20' room in less than 2 hours. Making a liberal allowance of 30 minutes for shifting, 4 rooms are underholed in a day with the employment of 2 men. This in a standard vein (4' thick) corresponds to 45 long tons, or at least to 7 men's kirving. As a matter of fact, the output is more nearly 70 tons per 10 hours. On this, with 2 men to a face, 11 rooms would have to be kept open to equal the supply of one machine. A mine producing 1000 tons of screened coal (1300 tons of "run of mine") could obtain it from 28 machine-rooms, or 77 hand-rooms. The operations are therefore more concentrated,—less territory has to be kept open than in hand-work; and this is an important feature, recommending the "iron man."

In the comparative cost of coal produced by the percussion-machines and that by the rotary-bar or chain per ton of run of mine coal on the 60-foot scale, it was found that a ton of run of mine, exclusive of power, costs 27.7 c., while a ton produced by pick-mining costs 39 c.; by the rotary-bar and chain machine, exclusive of power, the cost per ton was 29.7 c. as against 39.6 c. in the same mine, produced by pick-mining. Though this comparison indicates the apparent advantage in the use of projectile machines, it will be largely reduced when the number of tons which each type of machine produced daily is taken into consideration. The amount of slack, dirt, sulphur, etc., which is allowed for in machine-mining is, as a rule, 30 per cent, and on this basis machine coal costs 10 c. less per ton than hand-mined coal. The amount of lump coal produced per ton of run of mine is fully

6 per cent greater in machine than in hand mined coal. Its groove is only $2\frac{1}{2}$ " , as against an average of 6" for even the skilful miner, whose kirv is 9" high at the face and 2" at the rear. Over 3 cubic feet of coal is thus gained by the machine per yard of face, with less waste and more large coal. The ratio of slack produced by machine and by hand is $\frac{1}{18}$ to $\frac{1}{4}$.

The per-ton amount of powder consumed in 30 machine-mines is less than that consumed in 503 hand-mines during 1895 in Illinois, in the ratio of 113.7 tons per keg of powder to 35.6 tons. As to the output of the machine, it may be said to be an average annual of 12,000 tons, with two men as runners, ten as followers, engaged in the various occupations of timber-men, track-men, loaders, etc. The Harrison percussion-machine is expected to make eight entry cuts per day. In a coal-seam seven feet thick, worked by the pillar-and-room method, the monthly output was 78.93 tons per man, counting all men who perform the ordinary functions of the miner; an all-hand-labor mine in an 8-foot free coal-seam produced 69.55 tons per man per month. In a 4.5-foot seam, free from impurities, the reciprocating machine averaged 28 tons of lump coal daily, which equals 42.5 tons of run of mine. This represents in this vein an average undercut of about 170 square feet. A plant running 12 machines, and in all the standard makes, will cost about \$12,000. In operation one machinist will tend to and keep in repair 30 machines, and one blacksmith can sharpen the tools, etc., required for about 400 tons daily output. In Illinois the cost of maintenance of one machine is not far from \$20.00 per year.

In Hocking County, Ohio, during 1894 but one fatal accident occurred in producing 1,453,391 tons, 73 per cent of which was mined by machinery. Whenever coal is mined by machine and the records are available, the facts demonstrate a marked decrease in the accident list, and this is due to the fact that no men are exposed to the danger from fall of coal, and but very few to that from the caving of sides or unsupported roof. Again, the men selected to operate the machine and to prepare the coal usually are of the higher order of

intelligence found among the craft, and the other duties of the miner are assigned separately to the men, thus giving to each a specialty in which a greater or less experience is acquired. Accidents from powder-explosions, too, are very rare. The fact that 12.8 men on an average attend each machine, is a sufficient answer to the complaint of their replacement by machines; for, while less men may be actually employed in cutting the coal, a larger number of extra men would be employed attending to the machine, laying rails, putting up timber, blasting, loading, etc., and for the increased output obtained from the machines, as compared with manual labor. The one class of common laborers required is that of loading coal on the cars, each machine requiring four to eight. Again, as to the effect of the machine upon the miner's wages, experience proves that the introduction of machines has always led to an increase and not a decrease in the amount earned by the miner; and this, coupled with the easier conditions under which his labor is applied, must appeal to him strongly. The disadvantages rated against the machines, are: (1) waste of coal, (2) larger cost of plant, (3) necessity of a thick seam, (4) necessity of careful and skilful supervision.

The introduction of machines decreases the number of strikes, decreases the number of delays of standing shots, requires more systematic development of the mine, giving a steadier output, and concentrates the operations because less territory has to be kept open for the machines whose output exceeds that of hand labor, than in mines worked by hand only.

To keep seven uninterruptedly at work, there are necessary from three to five additional ones.

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CHAPTER 1X.

THE COMPRESSION OF AIR.

93. Theory and principles; heating during compression; influence of altitude; losses in the compression; equalizers and compound cylinders; construction of the machine and its requirements; means for rendering the resistance of the piston uniform. 94. Calculation of the work done upon the air; tables; formulæ; discussion of the valves and forms of the principal air-compressors on the market; air-receivers and their form and utility. 95. Conduction of the air; air as a motor; pipes, expanders, etc.; theory in the operation of the motor; tables of losses by friction; discussion of the economy of working with or without expansion. References.

93. ANY of the machines described in the previous chapter may be run by air or steam; but though steam is the cheaper motor, air has the advantage of giving cool, dry, ventilated rooms.

When air is subjected to pressure, its volume is proportionately diminished (see page 242), and, transported, its expansion is capable of being applied as is steam. To secure 100 lbs. of absolute pressure from "free air," its volume must be reduced to 0.147 the original bulk; for 200 lbs., 0.074. To obtain the same pressure from steam, it must be superheated to 338° F. and 388° F., respectively. By "free air" is understood air at the atmospheric pressure of 14.7 lbs. per square inch, absolute. By absolute pressure is meant the pressure above a vacuum, distinguished from gauge-pressure, which is measured above the atmosphere (14.7 lbs. absolute).

Owing to the molecular repulsion in gases, a compression of volume cannot take place without a corresponding development of heat, the increment varying with the initial tempera-

ture, as the accompanying table shows. The last column gives the factor of ratio between the initial and final absolute temperatures, t and T , of the air. $(t + 459)f = T + 459$.

Gauge.	Volume isothermic.	Volume adiabatic.	Temperature, Deg. Fahr.	Temperature, Deg. Fahr.	Factor.
0.0	1.000	1.000	61.	90.	
14.7	.500	0.612	175.6	211.9	1.222
29.4	.333	0.459	255.1	294.2	1.375
44.1	.250	0.374	317.4	362.0	1.495
58.8	.200	0.319	369.4	417.0	1.595
73.5	.167	0.281	414.5	464.8	1.681
88.2	.143	0.251	454.5	506.8	1.758

If this compressed air be immediately used in an engine, it will return to its initial stage of temperature and pressure. This expansion is said to take place adiabatically,—freely, without receiving heat. At 461.2° F. there is no pressure.

The above table is based on a sea-level pressure. At different altitudes the absolute pressure and density vary as below.

Altitude.	Pressure.	Density.
Sea-level.....	14.7	1.00
$\frac{1}{4}$ mile above	14.0	0.96
$\frac{1}{2}$ " "	13.3	0.91
$\frac{3}{4}$ " "	12.7	0.86
1 " "	12.0	0.82
$1\frac{1}{4}$ " "	11.4	0.78
$1\frac{1}{2}$ " "	10.9	0.74
2 " "	9.9	0.67

The increase in temperature is an obstacle to rapid running; it tends to expand the volume so heated, as indicated above. If, however, the expansion is resisted, the heat reacts on the compressed air and increases its tension and, consequently, its pressure. The increase of resistance to com-

pression is 0.00204 of the pressure for each degree Fahrenheit. A volume of air which has been compressed to 73.5 lbs. has a temperature of 414.5° F., by reason of which it would expand to 1.78 times its original bulk, in accordance with the following formula: $491u' = u(491 + T - t)$, in which the volumes u' and u correspond to the temperatures T and t . Since, also, it is impossible to contain the heat, spite of every precaution, and there is no necessity for its retention, it is extracted as soon as possible. The loss of heat begins with the instant of its development by radiation from the conducting-surface of the cylinder, receivers, and pipes. The cooling attachments added to the cylinder more or less perfectly complete the dissipation of heat and permit an isothermic compression. It may be noted that the increment of heat is greater in the early stages of compression than toward the final; so the cooling is best done at the beginning of the stroke.

Fig. 257 is a graphic representation of the adiabatic and isothermal curves of air. The volumes of air at various times

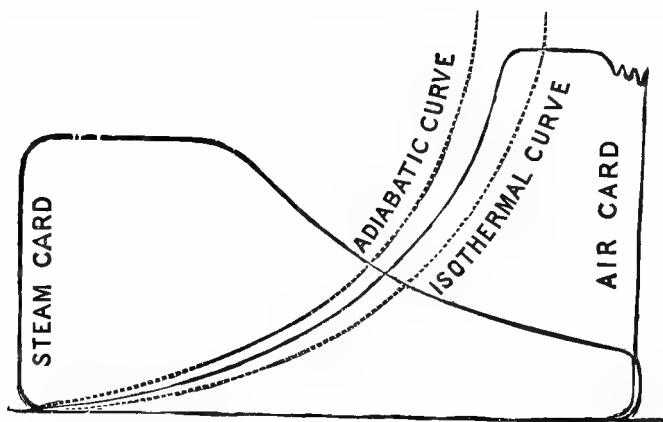


FIG. 257.

are laid off on the horizontal line and their corresponding pressures may be measured by the verticals. It will be seen that the adiabatic curve rises more rapidly than the isothermal or than the intermediate condition of cooling attained in the com-

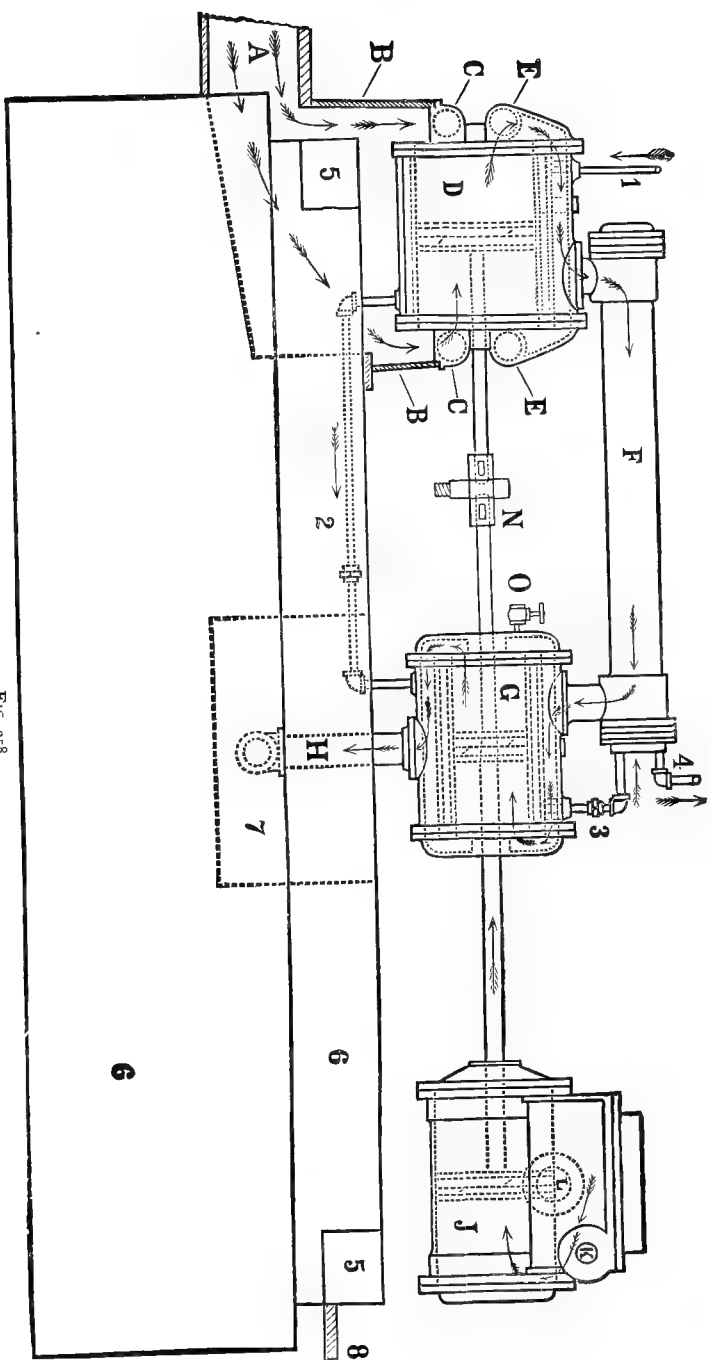


FIG. 258.

pressor. The indicator card also shows the behavior of steam when expanding from 58 lbs. pressure at 0.3 cut-off.

If the air cools, while its volume remains constant, a fall in pressure ensues, and the capacity for work in re-expansion is reduced. This is a serious loss, and is greater with the degree of compression. Rankine showed that the loss is rarely less than 65 per cent of the work performed by the motor. For 1, 2, 3, 4, 5, and 6 atmospheres, gauge-reading, the losses are 28, 37, 46, 50, 53, and 56 per cent of the original power. That is, the higher the pressure, the less is the efficiency of the expanding-engine. It is also greater if the cooling be not effected in the cylinder. Although the heat has been extracted from the air, it is still under pressure, and its unrestricted (adiabatic) expansion is yet capable of producing work; but by no means to the extent otherwise possible. There remains only the potential energy of the comparatively cool air, which is discharged at a temperature of about 180 F., amounting to somewhat over 6000 ft. lbs. per cu. ft.

Economic work is best obtained, then, by operating at as low pressure as consistent with the work. Again, if the air could be cooled before compression, so that after compression it will have the temperature of the surrounding air, better work would be done. The storage, too, of high-pressure air is difficult. The loss would be less if the air was heated, during its use, to an isothermic condition. This is impracticable, ordinarily.

Thus it becomes essential that the engine have its greatest power during the early part of its stroke, and yet drive the air-piston, with its maximum resistance at the end. For high pressures the difference is very marked. Single-acting conical cylinders have been used, but the compound-air cylinder has proven more effective for this purpose. In Fig. 258, the air, after a partial compression in *D*, is forced into the small cylinder *G*, where the operation is finished. The arrows show the directions followed by the air. This renders the resistance more uniform than where the compression is effected in one cylinder. A fly-wheel, and heavy parts, are partial equalizers,

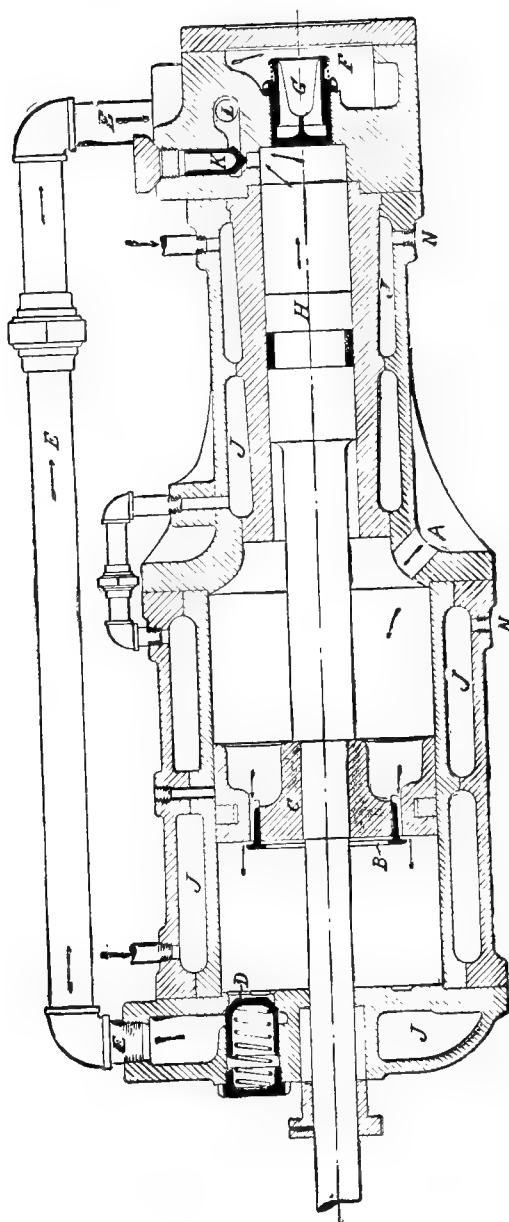


FIG. 259.

adding to the power when the steam is weak by expansion. The ratio between the air-resistance and the steam-pressure is fixed by the relation between the areas of the steam- and air-cylinders.

The air-cylinder is connected, directly or through intermediate gear, with the steam-cylinder, or geared to a water-wheel. It is simple or compound, and single or duplex. The air-cylinder is said to be "tandem" to the steam-cylinder when their pistons are extensions on the same rod; and "crossed," when alongside and joined to a cross-head. Its operation is identical with that of the steam-pump, and, substituting air for

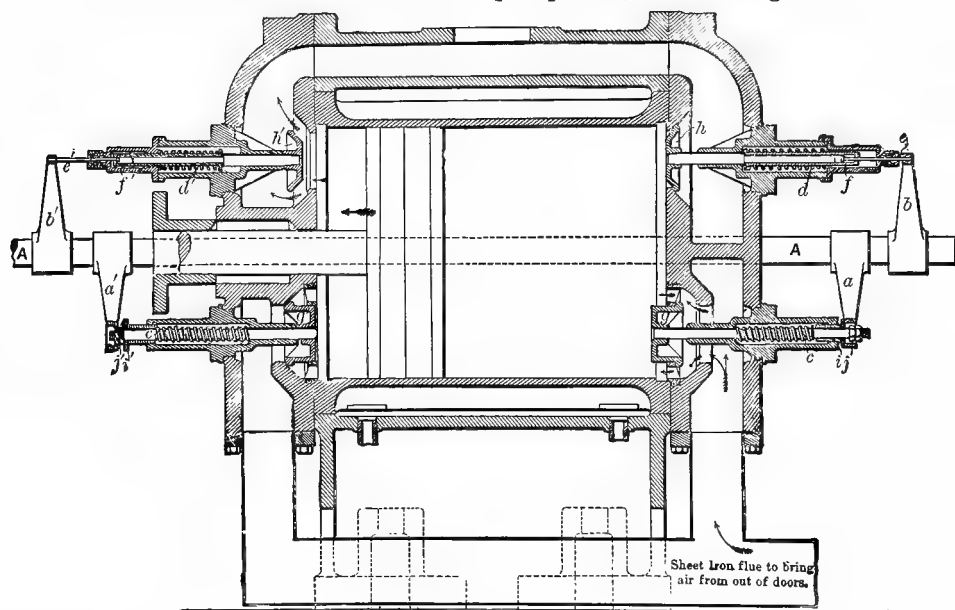
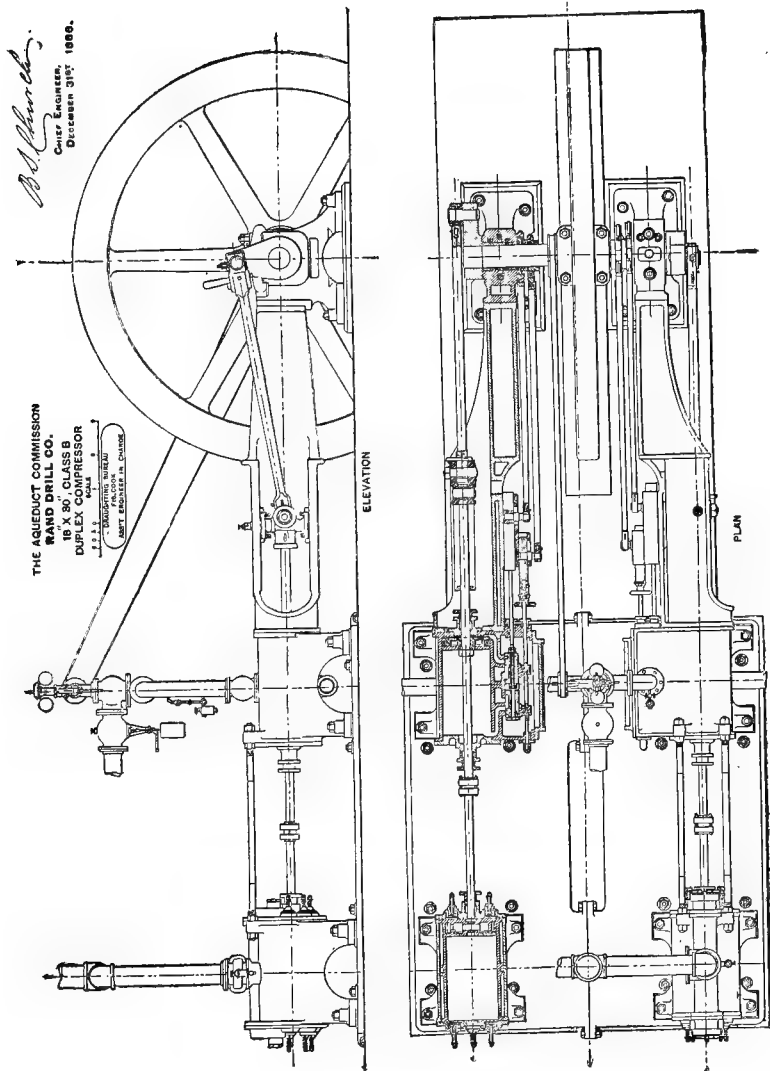


FIG. 260.

water, and dispensing with the air-chamber above, Fig. 93 may well represent the air-compressor.

94. The absorption of the heat of compression is accomplished by a cold-water jacket surrounding the cylinder. Formerly, a spray of water, injected into it, extracted the heat; but owing to the obstruction of the machines by the formation of



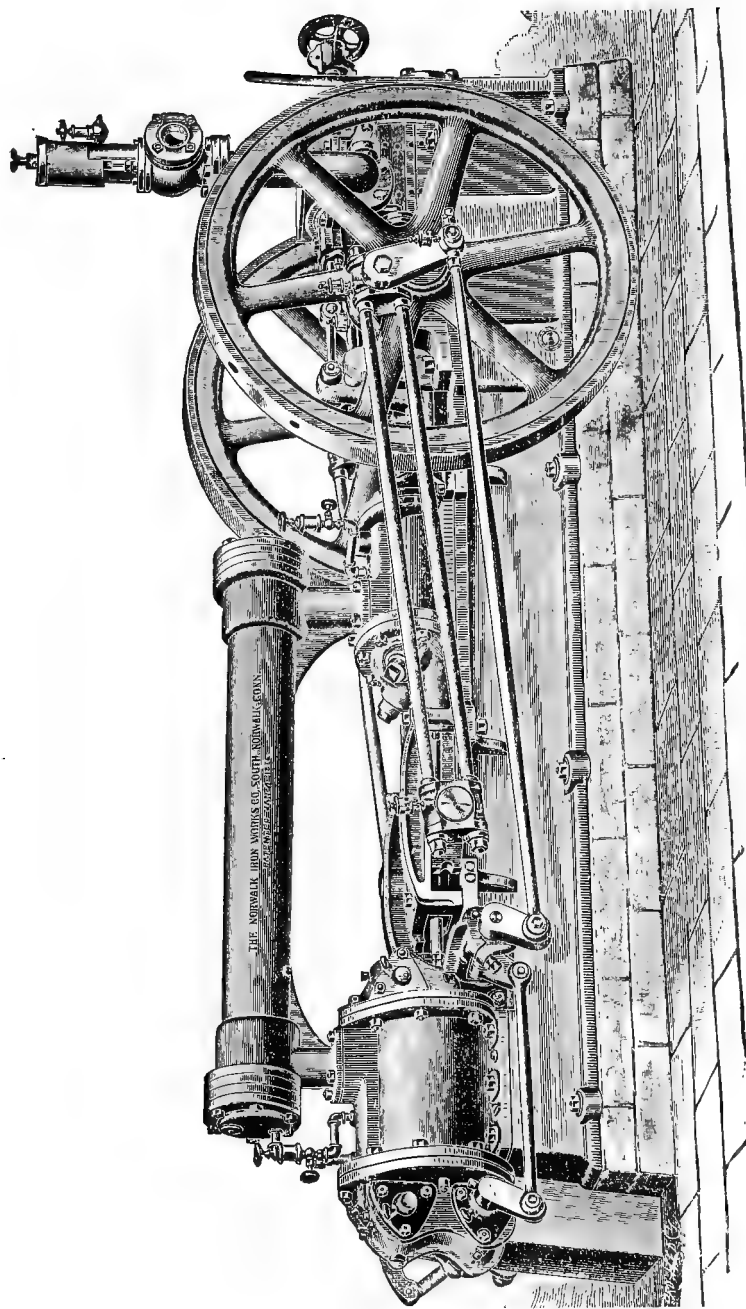


FIG. 262.

snow, in use, the plan was soon abandoned. The accompanying figure (260) is a section of the Rand cylinder, showing the cooling-jacket space, outside of which is the air-inlet. Fig. 261 illustrates the direct-acting duplex compressor, one half of the plan, being in section. Fig. 262 is a view of the Norwalk pattern, of which Fig. 258 is a section.

The cooling-water should be taken very cold, and the cylinder-lining should be a perfect conductor of heat in order to extract an appreciable amount of heat for the time of contact between the water and heated air is very short. In compound cylinders the air meets two currents; but even they cannot perfectly cool the air. So, often, an additional cooler is used, *E*, Fig. 259, in the Ingersoll, and *F*, Fig. 258, in the Norwalk. A reservoir filled with thin brass pipes circulating cold water offers a very efficient cooling-attachment, which saves as much as 10 per cent of the power, and about counterbalances the friction of the machine. Perfect cooling of 60 lbs. air would save at least 20 per cent of power.

The work of isothermic compression during one stroke of the piston is measured by the formula

$$W = P(V + v) \text{ hyp. log } \frac{V + v}{V' + v} - (P' - P)v ;$$

in which V and V' are the volumes of gas at P and P' lbs. tension per square foot, and v is the volume of the clearance. The work of adiabatical compression per pound of air is $183.43(T - t)$, T , t being the initial and final temperatures, measured in degrees Fahrenheit. It requires about 0.19 per cent more of work for every additional 1° F. of warmer air. One pound of free air at 90° F. raised to 88.2 lbs. pressure and to 464.8° F., requires 68,757 ft.-lbs. of work; of air at 60° F. to the same pressure, requires 67,171 ft.-lbs. For this reason, and that its density is greater, the air should be taken from outside of the engine-room.

Moreover, the air should be dry and free from dust, to prevent clogging of the machines, though the work of compressing a pound of dry air is somewhat more than that for moist

air, as the accompanying table shows; so, too, the temperature of the dry air rises more rapidly than that of the moist.

Absolute Pressure.	Temperature.		Work on One Pound.	
	Dry.	Moist.	Dry.	Moist.
14.7	68° F.	68° F.	—	—
22.	133.8	94	13,300	13,200
29.4	185.9	111	23,500	22,500
36.7	229.5	124	30,500	29,000
44.1	266.7	135	37,000	35,000
51.4	300.2	145	43,200	40,600
58.8	330.1	153	48,500	45,000
73.5	383.5	167	58,500	52,500
88.2	428.9	179	67,160	60,000

One inevitable source of reduction of cylinder-capacity is the clearance-space between the piston and the cylinder-head at the end of a stroke. The warm air filling it is never discharged, but, on the return stroke, expands and fills a volume that should have been occupied by the fresh atmospheric air. An increased length of stroke, or compounding the cylinders, are the only remedies. At 75 lbs. pressure, the single cylinder must be over three times as long as would the compound for equal clearance loss. Manufacturers dare not plan for a smaller than $\frac{1}{16}$ " clearance. A German device for obviating the effect of clearance consists in effecting a communication between it and the other parts of the cylinder. Indicator-cards show an increased efficiency of 5 per cent by the use of the contrivance in connection with a dry slide-valve.

The valves of the compressor are of the poppet, spindle, or ring patterns. Whatever their form, they should open quickly, have a full lift, and be ample in size. The inlet valves offer little difficulty, though for a short time they are subject to full reservoir pressure. An unrestricted entry for the air is obtained easily by the use of poppets held by springs. The Ingersoll-Sergeant compressor admits air through a hollow piston and rod (Fig. 232), and thus leaves more room for the discharge valves and the cooling-surface. The Norwalk-employs a Corliss valve. The inlet valves of the Rand are shown at *g, g'*.

Fig. 260. They are provided with guards that prevent falling into the cylinder.

The valves should be positive, and this the poppet obtains, though the tendency to "chattering" is the serious objection to it, particularly for discharge-valves. This arises from the two opposing efforts—one, of the air, to open, and the other, the spring, to close the valve. The Norwalk-Corliss pattern does away with this trouble in the high-pressure engines; as also does the valve-gear shown in Fig. 233. The arms, *a*, *b*, relax the spring-pressure and allow of the valve rising full lift without dancing. Poppet-valves can hardly be improved upon for low pressures, though their springs lose elasticity and open too soon. This reduces their efficiency, as also does any slip of the valves. In the Norwalk pattern (Fig. 235), a positive discharge is obtained by moving the valve by cams, such that it remains at rest till the pressure is sufficient to open it quickly. A difficulty about this, it would seem, is that, as the reservoir pressure constantly varies (unless perfectly regulated), the the construction of valves should receive careful attention. This has been corrected by an automatic movement given to their compressor-valves, which open at different points in the stroke, as desired.

The discharge valves require careful construction, for their leakage is equal to a large clearance-space. In order to reduce the friction of the air passing they are made large, and, to prevent inordinate loss and wear, as numerous as possible. An excess of engine-pressure over receiver-pressure is necessary to open the valves and expel the air. This unavoidable loss has an important bearing upon the uniformity of speed. An automatic regulator assists this to a certain extent, though, as a matter of fact, a hand regulator is found to be equally satisfactory for a long line of pipe. There have been devised plans for unloading the engine, and maintaining a uniform pressure, even under a heavy draft upon the receiver, but of their performance we have yet no returns.

Thus the simple principle of the air-compressor becomes exceedingly difficult of execution in an efficient manner. To obtain a compact, high-speed, uniform, rapidly cooling, efficient engine is not easy. These essentials are secured in various

ways in the Burleigh, Clayton, Delamater, Ingersoll, Norwalk, and Rand patterns. The engines are "straight-line," direct-acting, or duplex, supplied with fly-wheels, and run by plain slide-valve or Corliss engine, water- or electric motor. The direct-acting and horizontal form is preferable, for many reasons, though one objection to the straight-line form is its liability to centering. Fly-wheels—and some are weighted at certain points—remedy this somewhat.

High piston-speed is advantageous for economy of steam and capacity; but rapid wear, the difficulties of large valve-area, and the inordinate resistance developed, forbid a greater velocity than 300 or 400 feet per minute, except in the larger sizes. Automatic valve-gear, on the Rider system, are also added now; they entail no loss of steam, and, with the variable cut-off, regulate the engine speed. High-test cylinder-oil is required for lubrication; graphite is the best lubricator.

Duplex engine connections are by cranks at quarters. The frames are solid and well founded. For purposes of transportation to remote camps, they may also be had sectional. Water-power and wheels are admirably adapted for this work. Two 66" Swain turbines, with 16' fall, run four compressors 24×60, furnishing air for 20 drills, 8 hoisters, and 17 pumps, at the Republic mine. At the Anaconda, the 30×60 duplex, with Corliss valves, is the largest compressor that the author knows of.

The Burleigh is upright, and its peculiarity lies in the comparative sizes of the cylinders, and in admitting steam one-eighth of a stroke before the air. Its air-cylinder is single-acting. The Clayton has the usual poppet-valve, and is a compact machine, with its fly-wheel centrally located. The Delamater has an important contrivance for dropping the discharge valve from its seat. This form is very heavy. The Ingersoll, Norwalk, and Rand are the popular pneumatic machines. The Waring has a bonnet, or conical valve, like that in Fig. 233. Its pistons are moved by a rocker on the fly-wheel, the steam-cylinder being set at an angle to the horizontal air-cylinder.

The discharged air is stored in a receiver, whence it is withdrawn as required. It is simply a strong iron reservoir, of any convenient shape, and commodious enough to meet any draught upon it.

95. The air is conveyed to drill, coal-cutter, hoister, etc., by pipe, and in its transmission a great loss is experienced by reason of the friction.

Recurring to the formula in I, **50**, it will be recalled that the frictional resistance is directly proportional to the square of the velocity of the flow, directly as the length of the conduit, directly as its periphery, and inversely as its area, or the square of its diameter. As the periphery is proportional to the diameter the resistance becomes an inverse function of the diameter. Tables are given by manufacturers of the loss of pressure by flow in pipes, and it will be found therein that air at 32.8 feet per second loses 8.26 lbs. pressure in a mile of 10-inch pipe, 10.04 in an 8-inch, and 20.08 lbs. in a 4-inch pipe. Below is appended a table giving the loss in pressure, p , for various velocities, v , in feet per second, and different diameters of pipe, length being 1000 feet; q is the volume of "free air" passing per minute (air-cylinder capacity), corresponding to an assumed gauge-compression of 60 lbs.; q' , the volume at 80 lbs. This is copied from the Norwalk Iron Co.'s book.

v	1"			3"			4"			6"			10"		
	p	q	q'	p	q	q'	p	q	q'	p	q	q'	p	q'	q'
3.28	.143	6	7	.046	48	60	.034	86	100	.023	103	244	.014	537	680
6.56	.640	12	15	.209	96	121	.152	172	217	.104	386	488	.064	1073	1359
9.84	1.454	18	22	.488	144	182	.360	258	326	.244	579	633	.145	1610	2039
13.12	2.562	24	29	.838	193	243	.628	343	436	.419	772	977	.256	2146	2719
16.40	3.934	29	37	1.317	241	304	.982	429	544	.658	965	1221	.393	2683	3390
19.68	5.422	35	44	1.808	289	364	1.356	515	653	.904	1158	1466	.542	3220	4079
26.24	10.248	47	59	3.352	386	486	2.513	687	871	1.676	1544	1954	1.024	4293	5438
32.80	15.738	59	74	5.270	480	607	3.928	859	1088	2.635	1931	2443	1.573	5367	6798

Elbows and short turns will reduce this pressure. For any compression m than that given above, the bulk of the air *after* compression will be different from that at 60 by 74.7 : $m + 14.7$, taken inversely; or from that at 80 lbs. by

94.7 : $m + 14.7$. To carry this the velocity of the flow is different, also its friction. Suppose 300 cu. ft. of air at 60 lbs. per minute was required at 5000 feet from the compressor, the corresponding volume of free air is 1530 cu. ft. A 10-inch pipe would carry this with a loss of but 0.657 lb., a 6-inch losing 8.362 lbs. This would require a receiver pressure of 61 and 69, respectively. The 10-inch pipe would save 9.5 horsepower.

The power to be transmitted varies as the product of the air-pressure and its volume, which varies with the product of the velocity and the square of the diameter of the tube. Hence we may increase the power and diminish the resistance by enlarging the tube. The size most expedient is determined by the relative and absolute costs of power production and the pipe laid. The motor pressure must exceed that of the desired pressure at the expanding machinery by the amount of the friction loss. As the friction increases so does the most economical pressure. With a pipe friction of 5.8 lbs., 30 lbs. at the compressor will give an efficiency of 63 per cent; with a loss of 8.8 lbs., the compressor gauge-reading should not be below 38 lbs.; with 11.7 lbs., an efficiency of 56 per cent is obtained from a 47-lb. gauge-reading; while, if the friction should reach 26 lbs., the lowest admissible gauge-pressure is 82 lbs., when an efficiency of 50 per cent is to be obtained. This efficiency may be increased by diminishing the friction, but it can never be 100 per cent. The friction loss may be reduced by increasing the pressure of the air, thereby reducing the volume to be transmitted, and hence its velocity. But in doing this we have not increased the efficiency. So there is not likely a gain in employing high pressure, though this is rather a matter of local conditions. See table, page 506.

In utilizing air-power, pipes as large as convenient should be employed: 2 inches is as small as practicable, and this size is only for rooms and stopes; they should be united to not less than 4 inches in medium-length entries and levels.

All bends should be gradual; with leakage they reduce the pressure, which can be maintained only by care and judgment.

Unless purposed for ventilation, leaks should be plugged. Compressed air escapes at a velocity of over 20,000 feet per minute, and the consequent loss is apparent.

The pipes used are steel-riveted or lap-weld, as illustrated in Figs. 60 and 64. The joints should be carefully tightened. Means must not be neglected for providing for changes in length due to alternations of temperature. Iron expands 0.000007 its length per 1° F. This allowance is more essential above than below ground; and in shafts where the temperature is inconstant. Compensation-joints are necessary; at every 300 or 400 feet a copper U-tube is attached. Its straightening out will allow for contraction, while expansion of the pipes will double it up. At the Republic mine brass-lined expansion-joints every 500 feet allow for 12-inch movement. They rest on gas-pipe rollers. The Chapin iron-mine has \$500-expansion-joints at every 680 feet of the 24-inch pipe.

Irrespective of the friction losses, the power possessed by the compressed air would equal the work done upon it if the extremes of volume, temperature, and pressure are the same, for the cycles of changes which the gas experiences are duplicates. But friction and cooling absorb so much power that the "duty" of the pneumatic engine is small. Besides, as the compressed air is at a temperature not much above that of the surrounding atmosphere, it must fall far below the normal when used. Being colder, its volume is less than that occupied by it in the compressing engine; hence, even between the same extremes of pressure a return cannot be expected equal to the work expended.

In the maintenance of the isothermic conditions during compression there lies the greatest loss of power. If the heat there extracted could be returned to the gas while in use an economy of power is experienced, as the gas would expand isothermally. The cylinder (in drill or coal-cutter) encased in a hot-water jacket might accomplish the same result. But this is impracticable under normal conditions and the air must in expansion therefore be used adiabatically.

On the next page is Mallard's table showing the merits of

working the compressed air at full pressure, P' , or a complete expansion, taken from H. S. Drinker's "Tunnelling." The initial temperature, t , is 68° F., and the final, T .

Pressure Ratios.	Fall of Temperature after Expansion.	Theoretical Efficiency.	Fall after Working at P' .	Theoretical Efficiency.
	Degrees F.	Per cent.	Degrees F.	Per cent.
2	102.	85.5	72.3	82
3	140.	80.6	98.4	72
4	170.6	78.2	109.7	67
5	192.2	76.8	118.4	63
6	208.4	75.8	123.8	60
7	212.8	75.1	127.4	57
8	234.9	74.6	130.1	55
9	244.6	74.2	132.1	53
10	253.2	73.9	133.7	51
11	260.4	73.6	135.3	50
12	266.9	73.4	136.4	49

From this we see that, working without expansion, the refrigeration is not so excessive, but the efficiency is low.

Air at 60° F. and 60 lbs. pressure falls to 26° F., and if used expansively to 100° below 0° F. A saving in power would be had if air could be used expansively, but on account of the intense refrigeration (even the dry air of the Colorado climate at 60° F. contains sufficient moisture to saturate it at 10° F., below which point the ice will form) this is impossible. So, usually, full pressure is maintained during the stroke. There is a slight gain from the presence of vapor in the air and also in the work of compression (see table on page 392), and were it not for the formation of ice we would have an economic condition.

As the efficiency of the driven machine is not much over 60 or 70 per cent, the indicated horse-power of the entire combination does not exceed 25 or 30 per cent of the original motor power.

These facts account for the limited use of this physical agent. Yet it answers well for many purposes, notwithstanding those unavoidable losses inherent to the gas, and it will continue to retain its place as a valuable accession to mining. Its ventilating power enhances its value, and improvements in mechan-

ism will increase its efficiency. A simple compressor with a simple motor has about 40 per cent efficiency; a compound compressing cylinder, 46 per cent; with a compound air-cylinder and a compound motor, 51 per cent. Its sole motive competitor is electricity, which may surpass it in energy, but which cannot furnish ventilation without additional appliances.

As examples of compressor-plants and their price we may quote the following: An 18×30 duplex, at 50 revolutions, and 8-inch pipe, will run 26 drills, the compressor alone costing \$4800; an 8×12 (costing \$1100) will drive 10 coal-cutters; a complete plant for operating three ordinary-sized mining drills would cost \$3800—for two extra drills, steel, etc., add about \$800; an 8-drill plant complete cost, f.o.b. at St. Louis, \$8400, and weighed 42,000 lbs. The Chapin iron-mine, L. S., has a plant 3700 horsepower, giving an efficiency of 27 per cent, that cost \$500,000.

EXAMPLES.—Required the loss of air in 4400 feet of pipe, 10 inches diameter, if 1960 cubic feet of 50 lbs. air is to be conveyed per minute. 9.9 lbs. ✓

Required the sizes of pipes from three connected pairs of air-compressors, at 60.8 lbs. gauge, 29.4 revolutions, producing 1,826,700 cubic feet air per 24 hours, for 15 drills, at a distance of about 2100 feet, 14 at 2600, 21 at 3100, and 5 at 3600 feet. 12", 10", 8", and 4": total loss, 6.89 lbs. pressure.

Required the coal consumption of the above system of compressors having an efficiency of 40 per cent.

An 18×30 duplex at 75 revolutions gives 205.7 cubic feet of air at 80 lbs. absolute. How many $3\frac{1}{8} \times 6\frac{1}{2}$ drills at 220 blows will it drive, allowing for 12 per cent pipe loss? 18. ✓

At the Hoosac Tunnel 7150 feet of 8-inch pipe lost only 2 lbs. of pressure of a 68-lbs. (absolute) air. What was the velocity and the volume of air passing?

11' 6" per second, and 269 cubic feet per minute.

Required the volume of 60-lb. air to drive a pump with 14-inch air and 8-inch water cylinder against 200 feet head. ✓
Piston-speed 100 feet. 430 cubic feet free air.

Required the number of units of work necessary to compress 80 lbs of air from 14.7 lbs absolute pressure and 68° F. to a pressure of 88.2 lbs. per sq. in.

Neglecting the value for the clearance, v , of the piston in its cylinder (in the formula on p. 499), then for isothermic compression the work performed is

$$W = PV \text{ hyp. log } \frac{V}{V_1}.$$

P , the initial pressure in pounds per square foot of surface, $= 14.7 \times 144$. V is the volume occupied by the 80 pounds of free air at 68° F. and a normal pressure. V_1 is the volume occupied by the compressed air which by the Mariotte law is one sixth of V . The hyperbolic logarithm of 6 is 1.79. Whence

$$2116.8 \times 1064 \times 1.79 = 4,031,520 \text{ ft.-lbs.}$$

If the compression be performed adiabatically the work performed is measured by the formula $183.45(T - t)w$, in which the extremes of temperature, absolute, are represented by t and T and the weight of the mass of air to be compressed by w . To ascertain the value for T , the following equations may be solved :

$$T = t \left(\frac{V}{V_1} \right)^{0.408} = t \left(\frac{P_1}{P} \right)^{0.29};$$

$$\log T = \log [461^\circ + t] + 0.29 \log P - 0.29 \log P_1;$$

$$\log T = \log [461^\circ + t] + 0.408 \log [V \div V_1].$$

For certain pressures indicated in the first column of the table on page 491, the corresponding ratios of adiabatic volumes may be found in the third column. In the sixth column are given the factors $T \div t$ for the corresponding pressures and any initial absolute temperature, t . Solving, as above,

$T = 930^\circ$ absolute $= 469^\circ$ F and the work of adiabatic compression;

$$W_1 = 183.45 (930^\circ - 529^\circ) 80 = 5,886,680 \text{ ft.-lbs.}$$

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CHAPTER X.

MINE EXAMINATION.

96. Examination and evaluation of mines; sampling and measuring the deposit; features to be noted; capitalization; "ore in sight." 97. General remarks regarding the treatment of ores; factors determining their value; deleterious substances; various milling processes; cost of mining; formulæ for mine valuation. 98. The mining-labor problem; variety of skilled labor employed; selection of men; necessity for regulations and their enforcement; conveniences, hygienic and otherwise; number of shifts and their length, mode of paying; necessity for reciprocity; day's pay *vs.* tribute system; contracts and the mode of letting; pay by the output or progress; dead work; leasing mines. 99. Retrospective. References

96. THE examination of a property with a view to purchase or exploitation will not be a difficult matter to one who is cognizant of all the local economic conditions, and is familiar with the geognesy of ore-deposits. It calls for the exercise of sound judgment and technical instinct which we endeavor to cultivate: these faculties are self-taught; observation and caution will strengthen them. Vigilance and prudence prevent many of the fatal mistakes to which hasty conclusions of the human mind are prone. You cannot foresee or predict in detail the future of a mine. Your only basis is the general canons. To apply these to the selection of a system and plant you must collect and investigate all data in any degree suggestive.

Proceed to a careful study of the survey maps of the district, for they will assist in your determination of the trend, frequency, extent, and size of the ore-shoots. Close investigation of any idle mill or abandoned works in the vicinity, and the commonly received opinions regarding the causes of their

failure; may prove valuable as a guide. Converse with the miners, as they are in possession of valuable details, and may be able to furnish the history of similar enterprises. Consult all convenient sources before visiting the mine, then "sort" out the prejudices, "screen" the ignorance, "jig" the balance by technical knowledge, and examine the "concentrates" for probabilities.

On visiting the property proceed systematically with the geological examination of the enclosing rocks, the character of the vein matter, and its irregularities of thickness or pitch. Look closely for faults or slides. Collate the data thus obtained, and determine if the deposit be gash or fissure, pocket or bed. Note the toughness and hardness of the vein matter, the security of its walls, and the width and softness of the ore-streak with a view to calculating the cost of mining. The dimensions of horses, partings, or barren ground and the cost of their removal should be considered; as also the quality and quantity of the other available materials for support.

Wherever the mineral is exposed, samples are to be taken for assay to ascertain its value and quality. Every variation in the dimensions or character of the ore body should be sampled in a line across the entire exposure. Each streak or bench may differ and require sampling. In every case describe in the note-book the appearance of the face, its location, and the width of the streak. Take the precaution to arbitrarily select the spots for tests, and collect a sufficient quantity of each constituent mineral of the ore for its separate assay. Obtain freshly blasted or picked material. Examine closely for any evidences of concealment by plastering or timbering up of poor faces. Take "grabs" from the cars or stock piles. It would not be amiss to sample the dump. A sheet of paper, cut into snips, is thrown to the winds. Grabs are taken where they alight on the dump. The size of the dump will convey some idea of the amount of the work done. Seal all bags immediately upon filling, and leave no opportunity for injection in them of strong solutions of mineral, or other "salting."

Take all measurements that will serve as guides in estimating

the area and extent of the vein or bed, the amount or value of mineral capable of extraction, and the amount and value of that to be left for filling, mined or unmined. Be cautious, and secure results as accurate as can be measured. In estimating the "ore in sight," it is fallacious to be too liberal. Include only the mineral in the blocks exposed on at least two if not three faces by the connected drifts, shafts, or stopes. Be careful, then, in the use of this term. It is often reactive. In computing the body of mineral remember that all is not sold. A portion is not mined out, some is buried in the waste, some discarded in sorting, and some lost in the concentration. The ratio of the volume sold, as per smelter and mill returns, and that of the work done, will prove a reliable guide. Likewise with coal: of its actual volume every acre will sell about 1000 tons for each foot of thickness of the bed. Mines only slightly developed are not readily estimated; but by comparison with the records of neighboring properties a rough estimate may be made, bearing in mind that in veins, a proximity to profitable mines is of less monetary value than in beds. From the calculation of the volumes of mineral and their several assay values, the property may be evaluated. While enormous reserves and untold wealth may thus have been found, a great deal of hard systematic work must be done before the mineral has been mined, shipped, and drawn against. Several mines may be quoted carrying large bodies of $1\frac{1}{2}$ ounces of gold, which are absolutely worthless because the ore is rusty. So with a 55 per cent sulphurous iron ore, a 46-inch coal vein with 5 partings, or a 20-foot zinc bed with pyrites. For this reason the term "ore in sight" should only represent its net value. The quantity of coal in an acre of bed is found by the continued product of 100, the number of inches thickness, and the specific gravity of the coal. For bituminous coal it is 1964 tons, and for anthracite 1775 tons per foot of vein-thickness per acre of area.

The cash value of a mine is that which will net a given annuity to the investors. The amount of this dividend should increase with the risks run. At best mining is somewhat pre-

carious because of the variability of its dependent elements, and a greater return is expected from it than from U. S. bonds. So the annuity should be 10 to 30 per cent in addition to the legal rate of interest. Moreover, the deposit not being inexhaustible, the life of the mine becomes a matter of mathematical calculation, which must needs enter our estimate. Against this depletion, and the wear and tear of the machinery, the sinking-fund must provide. The yearly contributions to it must create a new capital within the period of its life, which unfortunately is often limited by the impatience of the operators to extract the ore in the shortest possible time. The mine is not a permanent source of revenue, and for the period representing its life should return an adequate income.

On this account a partly opened mine is depreciated by the amount it has shipped, but, on the other hand, this development work has more or less demonstrated its value. Prospects are mere speculations, and their values are inversely as their chances, which must be carefully weighed. The probable continuity of the ore-shoot to a reasonable extent may be quoted, but only as a speculative guide. Every mine, till its value has been assured by a steady output, is a business speculation, and must be considered as such. It may only be compensated for by the high prospective rate of interest. True, the security is not invariably as good as under some other investments, but safety and income are complements—in fact, reciprocals of each other. Those demanding security must content themselves with a moderate interest. So be explicit and frank in your estimate, that your client may be assisted in judging of the risks as well as the returns. Better recommend it for the cultivation of mushrooms than indulge in any imaginative representations, or juggle with figures.

Investigate fully the condition and security of timbering, filling, pillars, and other supports of an abandoned mine with a view to the cost of their reinforcement. The cost of reopening it may exceed the estimated value of the deposit. The causes for its abandonment are numerous. It may have been due merely to the inflation of its stock beyond a fair dividing

basis, or to a lack of ready means of communication which time has improved. The exhaustion of means with which to prosecute work may have been the cause of shutting down. Discord among the co-owners is a very common cause of failure. Be careful in accepting the testimony of their books. The permanent-improvement account is often delusive, because it is frequently made to cover awkward expenditures. Take no cognizance of those entries appearing upon the permanent-improvement account dated subsequent to the moment of actual production.

If the results of the examination do not prove sufficiently positive, or the information regarding the geology or extent of the deposit be insufficient to satisfy the expectancy of a good annuity, duty to your clients requires a full, frank statement, overrating neither the difficulties nor the value of the prospect. A six-months' working bond and option may be advised, or the diamond-drill may be resorted to. Before recommending purchase or exploitation, be sure that the enterprise has a *raison d'être*. Then proceed to a consideration of the capital required.

A justifiable capitalization is one necessary for the proper equipment of the property commensurate with the prospective annuity. This depends upon the scale upon which the property is to be operated. The amount of exposure and the number of faces for attack fix the number of men that may be employed, and either they or the hoisting capacity, the output. For reasons of economy it is advisable to open large areas for attack. This regulates the quality and quantity of the product, and reduces the per-ton working expenses; but maintaining much open ground is expensive, and large capital is needed for the extensive equipment. The choice between this plan and conservative work depends upon the commercial results anticipated. Having figured the amount of capital requisite to properly complete the plans laid out, do not accept any compromise with the parsimonious efforts of the directors, and attempt to work on less than what is deemed necessary for a successful issue. It is simply inviting failure and disappointment. One canon must be distinctly laid down—that some

time must elapse before returns can be realized. Be not over-sanguine in prophecy.

97. All the material coming from the vein must be either valuable, worthless, or injurious, and a metallurgist's skill is requisite in figuring upon the disposal of the ore constituents. The ore is either milling, concentrating, or smelting. A simple ore may be hand-dressed to advantage, though often the gangue is acceptable as a flux, and is not removed with the deleterious matter. The average smelting ores are silicious, and a basic gangue is acceptable if it is not a sulphide. A bonus is paid for an ore with a basic excess. Pyrites injure the quality of coal, iron ore, galena, and gold ores. Coarse auriferous pyrites may be cheaply stall-roasted to advantage. Otherwise washing is the only means of elimination. Blende is objectionable in smelting, and if argentiferous, in roasting. It interferes with amalgamation in pans. Jigging will separate it, and that, too, clean enough to be salable. The roasting of blende and pyrites at the mine for the manufacture of sulphuric acid is suggested as a means of rendering all the constituents marketable without great additional expense. Do not forget, however, that some of the valuable mineral is lost in the operation, and in order that the proper system be adopted to minimize the loss, the assay value of each constituent mineral should be ascertained. Very frequently the blende of an argentiferous ore has all the silver. Concentration under these circumstances would be useless.

Gold and dry ores may be treated by lixiviation, or smelted, and a few with little zinc and lead are better treated by pan amalgamation. These ores are rarely concentrated. Iron oxides with from 25 to 60 per cent of metal are treated as ores. Those with titanium, phosphorus, or sulphur are rejected. If intermixed with clay or loam, they are washed and picked at slight expense. Coal is everywhere acceptable, and is mined with profit if the bed is over 30 inches thick. Whether for gas, coke, or fuel, purity and calorific intensity are prerequisites. Coals are classified by the fuel ratios of their fixed carbon to

volatile and combustible matter. Clay is frequently washed and dried.

Hence it follows that every new mining company has this problem to solve,—how and where to treat the ore. A smelter may be built, when the ore is unfailing in quantity, but a variety must be at hand. On a big scale, and in close proximity to fuel, it is a most successful method. A concentration-works may be built. Such a mechanical treatment is applicable to almost every variety of ore. An amalgamation or a leaching mill may be erected, but fuel must be cheap and smelting charges and transportation high before competition with smelters will pay. The unchangeable character of the ore is essential to the success of any mill. Before laying the foundation the mine must be fully opened to at least a two-years' reserve, as otherwise another sad mistake may be added to the long list of monuments to similar folly that grace our gulches. Nothing is so certain as the uncertainties of vein continuance. Instances are not rare of sudden changes—a lixiviating ore to a heavy lead-zinc mineral; galena to bornite; free-milling to smelting ores; etc., etc. Proceed cautiously in this matter, minimize the risks, and limit the capitalization. Many a company has been brought to an untimely end by the bane of mining—ill-advised surface improvements. Extensive plank-roads may be dispensed with till there is something to ship.

Several investigators have prepared algebraic formulæ which, after the substitution of the values for the variable factors in a particular instance, will give the price to be paid, or the capital required. Amadee Bruat, Miller, R. W. Raymond and Prof. P. H. van Diest have contributed to this line of mathematical research. While in the hands of a proper manipulator these equations may be satisfactory, it is not safe to entrust the novice with the solution of so intricate a problem as the evaluation of mines by an inflexible rule, into which cannot enter fully all the varied local conditions.

It is unfortunate that in this recital of facts the cost of mining cannot be quoted. The reasons are easily understood. In the Lake Superior region it varies from 94 cents to \$3.49

per ton of copper rock hoisted. Gold is mined and milled for from \$2 up. All the expenses of iron-ore extraction are not over \$2.10 per ton in some localities. The cost of coal-mining varies from 62 to 90 cents per ton. In one section of Colorado a \$25 lead ore pays handsomely, while a similar \$40 ore near by is unprofitable. No constant proportion exists between the labor and the other items. The ratio of dead-work to ground opened varies considerably with the anxiety of the operators and the proportion of gangue. Ordinarily, the cost of stoping per cubic foot will be one tenth that of drifting, a thirtieth of the sinking, and a fifteenth of the upraises.

98. Besides a familiarity with the practice in vogue among his neighbors, the engineer must have an intimate knowledge of human nature, that the mine-labor problem may be successfully coped with. The large number and variety of men employed, the selection of men and their treatment, are intricate questions, delicate of adjustment.

Besides the foreman, boss, or "viewer," who is the chief officer, parallel with the superintendent of the metal-mines, we have captains, "butties" or contractors, timbermen, shaftmen, masons, hewers or miners, trappers to look after the doors, trammers or "putters," drivers, engineers, etc. The adjustment of their pay and hours is a difficult matter, and is the primary cause of strikes and lockouts. The design should be to secure a mutual interest of miner and employer, in the getting of the maximum of ore in the minimum of time. Good miners are essential to the success of the property, and an ability to judge of their competency is a trait which only long experience can form. A good miner can strike right or left handed, knows the mode of carrying on the work without further notice, and a single-handed man is worth the best wages going. An inexperienced miner is of no earthly account. He will drift off from the vein into the country because it happens to be softer, or from ignorance he will shoot mineral and gangue together and necessitate extra sorting; his consumption of steel and powder will be excessive, or his shots will "pop;" he cannot hold well for the striker, or he

will bruise his mate ; and he is likely to leave a bald face for the next shift to break from. In all, the greeny costs more than he brings.

The men being scattered in little gangs throughout the mine, the whole force cannot be under the superintendent's eye, and the grossest dereliction of duty may escape his notice.

For these and other reasons a uniform per-diem wages is an unsatisfactory solution to the problem of pay. The old hand certainly should command more than the tenderfoot. The quality and quantity of his work deserves better remuneration, and yet trouble is engendered by attempting to grade the day's pay of laborers on the same class of work. No one but a saint would undertake this.

Some form of contract system remains as a solution to this problem. Dead-work can readily be arranged for at a measured rate of pay. Contracts by the foot or fathom can easily be regulated to mutual benefit ; and the plan of fixing a certain minimum and maximum of earning has universal favor in the Lake Superior region. The mode of letting contracts by "shift option" is common. Shift or gang No. 1 has the first bid, No. 2 the next, etc. ; after all have given their bid, a chance is offered to any one to underbid the lowest. Besides dead-work, the mining of rooms or stopes in ore of uniform grade and quality is also contracted out in this manner on short times. During the winter, when shipments are slow or impossible, the contracts for dead-work are best let.

In the case of contracts there is an incentive to labor, but the interests of the two parties are not identical. The employé has no more interest in the ore than has the day's-pay miner, and is liable to waste mineral. His sole object is to show as large a measurement as he can.

In coal, iron, or other bedded mines the pay is by the car or ton extracted, and the tramming may or may not be paid for separately. A certain face is let out to a butty and his men, who may work it in one or two shifts. This plan requires constant supervision to prevent the admixture of slate. A certain reasonable percentage of slate or clay is

allowable, but an excess over that forfeits the entire car. What with this trouble and the disputes over weights, the life of the "tipple boss" is not a happy one. The practice of offering a bonus to men who exceed a certain given output meets with happy results.

One difficulty with this class of contracts is that the timbering and track-laying are very apt to be inefficiently done. This is effectually obviated by active supervision, a subdivision of the different departments of labor, and their assignment to specialists. While no system can be devised to perfectly meet all cases arising from and under it, contracting, in one form or other, offers a stimulus to intelligent work and fosters habits of observation. This is particularly true in vein-mining, where feeders would otherwise be ignored, while their pursuit might lead to valuable finds.

In the Missouri flat zinc-beds is maintained a system of dividing the ground into plots of 200 feet square and leasing them to operators, who for a definite period of time extract the mineral therefrom and dispose of the ore to the highest bidder, through the owners, who retain a certain royalty and have supervision over the workings. If the work has been done properly, this plan conduces to the benefit of all concerned. Without this, adverse results may be expected. It is a profitable system where a large territory is to be operated, or where capital is scarce and immediate profits dubious. Theoretically, leasing is wrong if applied to mines owned by parties with sufficient capital for developing; for if its operation will pay the lessees, it ought to give similar return with company work. That it frequently does not, demonstrates that there is "a nigger in the wood-pile." It is true the lessees will gut the mine of all the ore and will do no exploratory work; but undeveloped mines, or tracts entirely in virgin ground, have only probabilities on which to base the terms of the lease, which should be liberal as to time and area. In developed mines these probabilities can be approximately computed.

This plan is not confined to the region mentioned. The

duration of the lease may be a few months or years. In England it continues with the life of three named persons, and terminates with the decease of the third. The period in any event should be commensurate with the amount of preparatory work to be done. A short term would seem to be unjust to the lessees, particularly if a rich strike is made; but this is fully equalized by the custom that justifies the abandonment of an unprofitable plot, without any forfeit. On the other hand, the articles of agreement often allow the company to order a stoppage of work whenever it may desire so to do. Such leases are recognized by their nature as speculative, and are given under terms that might be a hardship were it not that the corporation is constrained to act with great honesty.

The great difficulty with this system, like to the others, is in the adjustment of dues, company and miner rarely agreeing in estimate. Such differences are inevitable with any form, and a rigid policy must be enforced.

A similar form of leasing, known as "tributing," is adopted in its most characteristic form in many of our American mines, with beneficial results. After the preparatory works are run and the mine has been blocked out, the stopes are leased for a month or so on a stipulated royalty. Each gang is expected to mine with reasonable diligence, to stope up only, to maintain good timbering, and to deliver the ore at the level mouth. The company hoists and markets the product and keeps the mine dry, retaining a certain percentage of the gross values for the privileges. Other conditions, of timbering, smithing, and supplies, are imposed, according to locality. This plan works admirably, except for the trouble over settlements, and is acquiring universal favor.

The men are required to work over-hand, because then the timbering is within easy inspection. Under-hand is not permitted at all.

In beds and pockety mines, with ores variable in quantity and quality, this method is profitably pursued, if the manager is vigilant. Towards the end of the term of the lease, miners not infrequently plaster up the face to deceive him and obtain

a renewal on a smaller royalty. Again, much trouble is experienced in the miners on a poor stope or "pitch" helping themselves from a neighboring richer tract.

Mines working on a high-grade mineral usually fit up a room for the change of clothes. The amount of pilfering is thus reduced.

The length of shift varies from 8 to 12 hours. The latter is too long, and even a 10-hour shift accomplishes less than an 8-hour. This is so well recognized, that urgent work is divided up into three 8-hour shifts per day. Men engaged in sinking or in wet ground have either better pay or shorter hours. Large mines, delivering the men below in buckets or cages, lose too much time for short shifts, and they are run under the 10-hour rule. Of those supplied with man engines and ladders, the tally is taken below and some use long, some short shifts.

As to the number of shifts, this very important point is not easily settled. For a given output, two shifts require an area opened and a roadway maintained, of only one half that of a one-shift mine. In metalliferous districts, somehow, night-shifts are not in favor. Certainly, day's-pay mines require very active, conscientious oversight to accomplish as much at night as by day-shift.

99. Attention to these economic details is highly important. No mine can succeed without good miners and conscientious labor, yet this does not constitute the sole element of success. The vital point of the laborer's concern is wages, the proper adjustment of which requires skill, tact, and judgment. Then ability to judge of the quality and efficiency of their work is only acquired by experience and observation. Dereliction or incompetency in this latter respect may undo all the economy and care in the planning and execution of the engineering details. Under these circumstances, the larger the mine and the number of employes the shorter the time necessary to bankrupt the owners.

The designing and selection of the machinery is not by any means the most intricate or even the most important of the multifarious duties of the mining superintendent. Com-

bined with the matters requiring technical knowledge are the endless details, including the supervision of ore sales, the management of men, and the deciding of disputes. A manager incapable of combating these emergencies simply tempts ill-fortune and invites disaster.

The very nature of ore occurrences is such that the element of chance must needs figure in mining as it does in other business, but with the employment of an equal judgment and discretion the result should be equally satisfactory.

The selection and choice of a competent manager is not to be made hastily. The blunder, so frequently committed, of sending a clerk or relative from the counting-room to "run the mine" is responsible for the inevitable failure. Not only is he ignorant of the principles of mining, but he lacks sympathy with his surroundings; nothing recommends him for the position, except, perhaps, his consanguinity or his integrity. The selection of an excellent foreman may counterbalance some of the error in the management, but there is usually nothing in common between the foreman and the manager except the question of salary, which in the case of the former is meagre compared with that of the superior officer, who has little work or experience. The indifference that ensues soon becomes manifest in all branches of work, and the manager has no remedy until the funds become low, when he disappears from the camp, leaving odium upon himself and his class.

Frequently the same superintendent has launched out into that bane of mining work—premature surface improvements, palatial residence, ill-advised mill or process for treatment, and such monuments of folly as should stand out as warnings to succeeding corporations; but the same old mistakes follow one another closely, and striking examples of contrasting extremes are easily quoted.

Until the value of the lode has been demonstrated, neither mill nor elaborate improvements should be erected, for up to that time the prospect is merely a business speculation, and may or may not prove a successful venture. Grass-root bonanzas are rare.

When the plans are being laid, the educated engineer who exercises the business sense required for any other manufacturing pursuit will adopt the tried and true processes: not necessarily those of the camp,—for custom is time-honored to men, and innovations are looked upon with suspicion, and resisted, not having the seal of local usage,—but the most improved methods of successful camps. To such careful, observing management the many mines of Europe owe their continued prosperity, after three hundred years of working. Of shrewd business methods, the Atlantic mine (Fig. 5), with its heavy dividends, 1890, from an ore yielding but 13.27 lbs. of refined copper per ton of rock stamped, is a notable example.

In 1894, Mr. Albert Williams, Jr., in replying to the mooted question, "Does mining pay?" remarks that it is pretty conclusively established in the affirmative, with certain qualifications. It does not, if prosecuted loosely and regarded as a mere gamble, and if its purchase or investment is not characterized by business precaution, or it is not operated by one of trained experience. "Many of its operations can be planned and estimated with nearly the same precision as bridge-building and railroad work."

By actual and costly experience, the "practical man" learns what the "theoretical man," the graduate, has been taught—that to profit by the experience of others is wisdom. System will replace obsolete, crude hand-to-mouth methods of yore, and many an idle mine may be quoted that one keenly alive to the improvements in mining appliances might convert into a prosperous property.

Such knowledge comes through the study of the ephemeral conditions through which mining has passed. A compromise between or a union of theory and practice, and in such manner as to inculcate the fundamenta of technical knowledge that will enable the engineer to bring the fancy of expectation to the level of the facts of experience, is the purpose of the School of Mines.

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APPENDIX.

SAMPLE EXAMINATION QUESTIONS

FOR APPLICANTS FOR OFFICE OF MINE INSPECTOR OR
UNDERGROUND MANAGER.

1. Why are conical drums necessary in deep shafts?
2. Explain the difference between *long-wall* and *pillar and room*, and the relative amounts of coal produced per acre.
3. In a four-foot seam, 80 fathoms deep, what size would you make the pillars, having regard to the ultimate extraction of the greatest quantity of coal combined with safety to the workmen?
4. Which requires the largest pillars, thick or thin seams of coal?
5. What size and width would you drive entries, gangways, and rooms; and what size would you leave ribs and pillars for safety and economy?
6. Would you mine coal by underhand? Why?
7. What effect does gob, if any, heaped against the side of a pillar have upon its strength?
8. Describe how you would employ the *filling* system for a 20-foot vein of soft coal, when it pitches at 80° from the horizontal.
9. How would you test a steam-boiler to ascertain its safety?
10. What kind of a hoisting-engine would you consider the most suitable for a deep coal-shaft?
11. Explain fully the advantages in a deep shaft of having a series of lifts instead of one long lift to the surface—for pumping-engines.
12. Describe some of the best forms of safety appliances.

13. How would you avail yourself of a voluminous water-fall at small head for utilizing the power?

14. Give some idea of an electric plant you would suggest for a copper-mine with 8 levels, 1200 feet deep, and 30 stoping faces. Ample power to be had.

15. What precautions would you take upon approaching an abandoned mine?

16. If the workings of a mine are approaching the abandoned workings of another mine, in which there is a head of water of 100 feet, how much coal would you leave as a safe barrier between the two? And if you should tap it with a two-inch hole, what would be the number of cubic feet of water discharged per minute?

17. In a seam having a dip and rise of 1 in 6, and the direction of the plane of the coal being to full rise, sketch what you consider a good form of *long-wall* working for it, having regard to the ventilation, direction of the drawing roads, etc.

18. Under the same conditions, give us a sketch of a *stoop-and-room* working by which the greatest percentage of the seam can be got out.

19. State particularly how you would lay off a coal-mine, so as to conform most effectually with the provisions of the laws, and make a small diagram of the same, showing location of ventilating apparatus, doors, air-splits, over-casts, and direction of air-currents from inlet to outlet.

20. How would you test the safety of the roof in entries and rooms?

21. What is the practical use of the barometer and thermometer in mine inspection?

22. What are the relative merits of fan, furnace, or other appliances for producing ventilation?

23. Can you with safety use a furnace to ventilate mines where explosive gases are generated? If so, describe the kind you would use.

24. How would you kindle a furnace situated at the highest opening, when the exterior air is at the temperature of 90° and that of the mine at 50° , and the current moving in the direc-

tion of lowest opening, to prevent filling the mine with smoke?

25. How does the steam-jet compare with the furnace as a means of ventilation, and how should it be applied?

26. Describe particularly the methods and instruments by which the velocity of air-currents in mines can be measured.

27. What safety-lamps do you consider the best?

28. Under ordinary conditions as to gas, what quantity of air would you circulate in a pit with 100 men, and what are the least dimensions you would have the air-ways?

29. Give your reasons for making air-courses as large as possible.

30. What would you call a safe velocity for an air-current in mines where explosive gases are given off in large quantities?

31. State your opinions regarding "splitting of air," and the advantages, if any.

32. In case of an explosion underground, whereby the furnace-doors, over-casts, and air-stoppings are displaced or destroyed, what method would you adopt to restore circulation to admit of a prompt rescue within?

33. What are the usual causes of fires in mines; how would you prevent them or suppress them?

34. How would you reopen a mine after flooding or after an accident?

35. Sketch what you consider the best form of a shaft 600 feet deep for an output 300 tons per day, including provisions for pumps, and showing the arrangements of sides and cages, with dimensions.

36. What are the difficulties encountered in sinking and timbering-shafts through pitching strata? How would you overcome them?

37. Explain how you would prevent water met with near the surface from getting into the shaft.

38. What explosives do you favor for a gaseous coal-mine?

39. What explosives do you prefer in soft galena ground?

40. Why is less explosive consumed in *long-wall* than in *pillar and room*?

41. Under what conditions would you recommend an installation of air-compressing plant with power-drills?

42. Under what circumstances would you prefer the diamond-drill for underground work?

43. Explain the best modes of drawing coal along a level road, or one not dipping sufficiently to take away the rope.

44. Under the usual conditions of tram-rails, what is the flattest gradient for a self-acting inclined plane, 300 fathoms long, to pass 100 tons in 8 hours? Sketch the best arrangement of it at the top.

GLOSSARY OF MINING TERMS.

Absolute pressure. The pressure reckoned from a vacuum.

Absolute temperature. The temperature reckoned from -46° F. or -273° C.

Adit. A horizontal passage from daylight into a mine, and *on* or *along* the vein. It differs from a *tunnel* in that it is not through country rock across *to* the vein, it differs from a *drift*, which has neither end in daylight.

Aerophone. A respirator in the form of a tank, receiving the exhalations from the lungs, which contains chemicals designed to revive the air and render it fit for breathing. It is used by rescuers after mine accidents.

After-damp. An irrespirable gas remaining after an explosion of fire-damp

Anemometer. An appliance for measuring the velocity of an air-current

Anticlinal. A fold of the rock or strata, convex upward.

Apex. The edge of a vein nearest to the surface

Back-pressure. The loss, expressed in lbs. per sq. in., due to getting the steam out of the cylinder after it has done its work.

Balance-bob. A heavy triangular truss, the long horizontal arm of which supports a weight at one end ballasted to balance the weight of pump-rods at the other

Barrier pillars. Pillars, larger than the ordinary, left at stated intervals to prevent the crushing of the roof from extending beyond the section enclosed by them

Bar-timbering. A system of supporting a tunnel roof by long top bars while the whole lower tunnel-core is taken out, leaving an open space for the masons to run up the arching. Under certain conditions the bars are withdrawn after the

masonry is completed, otherwise they are bricked in and not drawn.

Battery. The lower platform of a coal-chute. A term also used to define a timber bulkhead in a gallery.

Bearing-up stop. A partition of brattice or plank that serves to conduct air to a face

Bed. A seam of mineral occurring among the stratified rock.

Bench. The divisions of a coal-bed caused by seams of clay or slate; also used to express the artificial divisions in the process of mining; also a terrace at the outcrop of a seam.

Black-damp. Carbonic acid gas.

Blast. The operation of forcing air by blowing. The operation of exploding powder or other agents.

Blind drift. A horizontal passage in the mine not yet connected with the other workings.

Blind lead or blind lode. A vein having no outcrop.

Block coal. Coal that breaks freely into rectangular blocks.

Blossom rock. The rock detached from a vein, but which has not been transported.

Blower. A discharge of gas from coal; also a fan for forcing air into a mine.

Blow-out. The decomposed mineral exposure of a vein.

Bluff. Blunt.

Bob. See *Balance-bob*,

Brattice. A canvas or plank partition, nailed to posts longitudinally, with a level or shaft to divide the same into two compartments for the purposes of separating two air-currents

Break-through. A narrow passage cut through a pillar connecting rooms.

Breast. The face of a gallery or heading

Buggy. A small mine-wagon for conveying coal from face to gangway.

Bulling-bar. An iron bar used to pound clay into the crevices crossing a bore-hole, which is thus rendered gas-tight.

Bull-pump. A single-acting direct pump consisting of a steam-cylinder placed over the shaft. The steam drives its piston, to which the pump-rods are attached. By means of the piston attached to the rods the water is lifted by the steam-pressure. The down-stroke is effected by the weight of the pump-rods

Buntons. Timbers placed horizontally across a shaft. They serve to brace the wall-plates of the shaft-lining, and also, by means of plank nailed to them, to form separate compartments for hoisting or ladder-ways

Butt. The end faces of coal.

Butt-entry. The gallery driven at right-angles with the butt-joint

Butty. A miner working on contract.

Cage. An elevator platform.

Cap. A term signifying the point at which a vein is contracted; it is also the rock covering the ore.

Cartridge. A paper tube filled with explosive.

Casing. The tubing of a well-hole to prevent caving.

Centre. A temporary support, serving at the same time as a guide to the masons, placed under an arch during the progress of its construction.

Chain pillar. An untouched block of mineral on either side of the gangway

Choke-damp. Carbonic acid gas

Chute. An inclined trough or timbered shaft through which ore is delivered by gravity to a receptacle below. See also *Shoot*.

Clack. A pump-valve

Clearance. The space between the piston at the end of its stroke and the valve face, or the end of the cylinder.

Cleat. A joint produced by the natural tendency of coal or rock to cleave or split in a certain direction not parallel to the plane of bedding

Coal-measures. The strata embracing the carboniferous coals.

Column-pipe. The line of pipe through which the mine-water is pumped

Compression. In steam practice, the action of the piston in compressing the steam remaining in the cylinder, after the closure of exhaust-valves, into the clearance-space.

Contact vein. A vein lying between two dissimilar rock masses or strata.

Counter. A cross vein.

Counterbalance.—Counterpoise. A weight used to balance another weight or the vibrating parts of machinery

Counter gangway. One which is driven diagonally to the rise until the workings are reached, when it turns off parallel to the main haulage-way.

Country rock. The main rock of the region through which the veins cut; or that surrounding the veins.

Crab. An iron windlass for moving heavy weights

Creep. The crushing of the overlying rock resulting in the floor rising.

Crib. A framework built like a log-cabin. It may be a mere pillar afterwards filled with rock, or it may be the lining to a mill-hole or shaft.

Cross-course. An intersecting vein.

Cross-cut. A horizontal passage driven across the country rock to a vein (p. 21).

Cross-heading. A transverse drift which is driven for purposes of ventilation from one gangway to another.

Culm. The fine waste of coal-mines containing dirt as well as coal-dust.

Curb. A timber frame intended as a support or foundation for the lining of a shaft

Cutier. A term employed in speaking of any coal- or rock-cutting machines, the men operating them, or the men engaged in underholing by pick or drill.

Dam. A bulkhead

Day-shift. The gang of miners working during the day-time.

Dead. The valueless matter of a vein, also gangue or waste.

It is usually made the stowing or filling of an excavated portion of the seam, bed, or vein, and then is called gob, waste, or stull dirt. Sometimes the term is employed in speaking of a sluggish ventilating current.

Dead quartz. Quartz carrying no mineral.

Dead work. Exploratory or prospecting work that is not directly productive

Deposit. Irregular ore-bodies—not veins.

Dip. The angle, measured by the steepest line in the plane of a layer of rock, from the horizon.

Dirt-fault. A partial replacement of coal in a seam by clay. Not a true fault.

Drag. The point of union of two veins which meet without intersecting

Drift. Any subterranean horizontal passage. See *Adit*.

Driving. Excavating drifts, adits, or levels

Drum. The cylinder on which a hoisting-cable is wound.

Dumb drift. A gallery which conducts the air around a ventilating furnace to the up-cast shaft

Dump. The pile of rock which has been hoisted to the surface and deposited there. It may be said to be a low-grade ore reserve.

Duty. The unit of measure of the work of a pumping-engine expressed in foot-pounds of work obtained from a bushel, or 100 lbs., of fuel

Dyke. A fissure filled with igneous matter.

Empty or *empties.* An unloaded car or the track along which it travels.

Exploder. A chemical employed for the instantaneous explosion of powder.

Exploitation. The working of a mine, and similar undertakings; the examination instituted for that purpose.

Eye. The hole in a pick- or hammer-head for receiving the handle.

Face. The exposure of rock at which work is being done.

Fathom. A volume of rock equal to six feet square multiplied by the thickness of the vein.

Fault. The dislocation of a vein. The term is also improperly used in coal-seams. See also rock fault or dirt fault.

Fire-damp. A carburetted hydrogen gas, inflammable, and specifically lighter than air

Fire setting. The process of exposing very hard rock to intense heat, rendering it thereby easier of breaking down

Fissure vein. Any mineralized crevice in the rock of very great depth (p. 5).

Float. Broken and transported particles or boulders of vein matter.

Floor. The stratum below a mineral bed.

Foot-wall. The face of rock below the vein.

Forepoling. A mode of timbering

Free. A term employed in speaking of loose mineral.

Fuse. A tube, ribbon, or wire filled or saturated with a combustible compound, used for exploding powder.

Gad. An iron or steel wedge; a chisel-bit pick

Gallery. A horizontal passage.

Gallows-frame. The frame supporting a pulley, over which the hoisting rope passes to the engine

Gangue. The barren portion of the vein

Gangway. The principal level of a coal-mine.

Gash-vein. A mineralized fissure that extends only a short distance vertically. It may be confined to a single stratum of rock, but is a comparatively shallow vein.

Gateway. A gangway having ventilating doors.

Gauge-pressure. The pressure shown by an ordinary steam-gauge. It is the absolute pressure plus that of the atmosphere.

Goaf. The excavated space of a coal-mine, usually filled with the valueless portion (*gob*) of the seam

Gobbing-up. Filling with waste

Gouge. The layer of clay, or decomposed rock, which lies along the wall or walls of a vein. It is not always valueless.

Guide. The timbers nailed to the timbers of a shaft for the purpose of guiding the cage

Gunboat. A skip; a self-dumping box used in slopes.

Hanging-wall. The wall of rock above the vein.

Head-gear. A derrick.

Heading. A drift or air-way. The section of tunnel driven in advance of the lower section or bench.

Heave. A dislocation of the strata.

Helve. A handle.

Hewer. A coal-miner.

Hitch. A dislocation of a vein. Also, a shoulder or hollow cut in the rock to support one end of a stull or other timber

Hogback. See *Horse*.

Holing. The picking of a groove in the lower part of a coal-seam for the purpose of facilitating the breaking down of the upper mass

Horse. A mass of country rock lying within a vein. Any irregularity cutting out a portion of the vein. See *Dirt-fault* and *Rock-fault*.

H-piece. The portion of a column pipe containing the valves of the pump

Incline. An entry into a mine following the dip of the vein

Indicator. That which points or directs.

Some forms show the position of the cave in the shaft. Others record upon paper the pressure of the steam in an engine-cylinder at various points in the piston-stroke.

Inplace. A vein or deposit in its original position.

Intake. The entry which conducts the incoming air-current to the mine. It is synonymous with *downcast*.

Jar. That part of the drilling apparatus which takes up the shock of impact of the falling tools upon the bottom of the hole.

Kibble. An iron ore-bucket.

Kirving. See *Holing*.

Lagging. The slabs or small timber placed between the main timber sets and the roof or walls to prevent small rock from falling into the drift

Lath. A plank laid over a framed-centre or used in poling.

Laundry-box. The box at the surface receiving the water pumped up from below

Level. A horizontal passage in a vein-mine, numbered 1, 2, 3, etc., consecutively from the surface, or from the cross-cut tunnel, down.

Lift. All the mine workings connected with, opened from, and mined out at one level; also the length of pump-pipe between stations

Live quartz. A variety of quartz usually associated with mineral.

Location. A mining claim

Lode. A mineralized fissure

Long-wall. The system of mining coal without leaving any pillars.

Man-engine. A mechanical appliance for raising and lowering miners (see

Man-hole. A hole or an auxiliary shaft through which a man may pass in going from one level to another, into a stope, or from one ladder to another.

Measures. A term embracing the strata of a geological series.

Mill-holc. An auxiliary shaft connecting a stope or other excavation with the level below.

Mill-run. A test of the value of a quantity of ore as distinguished from an *assay*, which tests "pocket specimens."

Mineral. Any constituent of the earth's crust that has a definite composition.

Mineral oil. Petroleum or other liquid obtained from the earth.

Miner's inch. The unit of measurement of water used by the sluice-miners. It is that amount of water hourly discharged through an opening 1 inch square under a head of several inches. If the head is 7 inches and the hole is through a plank

2 inches thick, a miner's inch is equal to about 90 cubic feet per hour.

Mining. In its broad sense embraces all that is concerned with the production of minerals and their complete utilization.

Mining retreating. A process of mining by which the vein is untouched until after all the gangways, etc., are driven, when the mineral extraction begins at the boundary and progresses toward the shaft

Moil. A short length of steel rod tapered to a point, used for cutting hitches, etc.

Narrow work. Working places narrower than the rooms, entries, headings, break-throughs, gangways, etc.

Nitro. A corrupted abbreviation for nitroglycerine or dynamite.

Orc. A mineral of sufficient value (as to quality and quantity) which may be mined with profit.

Ore-shoot. A large and usually rich aggregation of mineral in a vein. Distinguished from *pay-streak* in that it is a more or less vertical zone or chimney of rich vein matter extending from wall to wall, and having a definite width laterally.

Outcrop. The exposed portion of a vein on the surface.

Outlet. An exit passage from the mine.

Output. The product of a mine.

Panel. The division of a mine which is isolated from neighboring districts and provided with distinctive haulage and mining systems.

Parting. A joint in the rock, or a crevice in a seam, filled with clay or slate; a switch or turnout to allow loaded and empty cars to pass one another.

Pay-streak. The thin layer of a vein which contains the pay-ore

Pike. A pick.

Pinch. A contraction in the vein.

Pipe. An elongated body of mineral. Also the name given to the fossil trunks of trees found in coal-veins.

Pit. A shaft.

Pitch. See *Dip*.

Pit-man. The shaft-man who attends to the shaft equipments, pumps, etc.

Placer. A surface accumulation of mineral in the wash of streams.

Plane. An inclined tramway for lowering cars by gravity or raising them by means of a stationary engine

Plat. A platform. A swinging or revolving door used intermittently to connect two trackways.

Plumb (adjective). Vertical.

Plummet. A string, or fine copper wire, attached to a heavy weight ; used for determining the verticality of shaft-timbering

Plunger. The solid piston of a force-pump

Pocket. A rich and large body of ore in the vein.

Poling. The process of timbering by the use of poles, for timbering in soft ground

Poll-pick. A combination pick- and hammer-head.

Poppet. Also *puppet.* A pulley-frame or the head-gear over a shaft. A valve which lifts bodily from its seat instead of being hinged

Post and stall. See *Pillar and Stall.*

Power-drill. A rock-drill employing steam, air, or electricity as a motor.

Pre-release. The act of discharging steam or air from the cylinder before the piston has reached the end of its stroke.

Prop. A piece of timber or metal placed normally to the roof or wall for its support.

Prospect. The name given to underground workings the value of which has not yet been made manifest. A prospect is to a mine what mineral is to ore.

Prospecting. The process of seeking pay-ore or the preliminary operations of a mine.

Pump. Any mechanism for raising water out of a mine.

Quick (adjective). Soft, running ground ; an ore or pay-streak is said to be quickening when the associated minerals indicate richer mineral ahead.

Rafter timbering. That in which the timbers appear like roof-rafters

Reacher. A slim prop reaching from one wall to the other.

Reamer. An enlarging tool.

Reef. The outcrop of a hard vein projecting above the surface. Also applied to auriferous quartz lodes.

Regulator. A sliding-door to apportion the amount of air to be admitted into a section of the mine.

Rib. A pillar of vein-matter left to support roof or wall.

Robbing. The taking of mineral from pillars

Rock-fault. A disturbed portion of a vein in which coal is replaced by sandstone.

Roof. The stratum overhead.

Room. A working place in a flat mine; corresponds to *stope* in a steep vein.

Run. A mode of contract work in which steep parts of coal-seams are driven and paid for by the lineal foot or yard of progress.

Saddle. The ridge of a stratum or ore-bed.

Safety-cage. One supplied with safety appliances

Safety-lamp. A lamp in which the flame is protected from immediate contact with the surrounding atmosphere.

Salting. Placing foreign ore in the crevices of a vein or elsewhere to fraudulently raise its apparent value.

Samson-post. An upright supporting the working beam which communicates oscillatory motion to pump or drill-rod

Sand-pump. See *Sludger*.

Scale. The incrustation deposited in boilers from evaporated waters.

Scraper. A tool for cleaning out drill-holes

Seam. A layer of mineral.

Seed-bag. A water-tight packing of flaxseed around the tube of a drill-hole to prevent the influx into the hole of water from above.

Selvage. See *Gouge*.

Sett or *Set*. A frame of timber.

Shaft. A vertical opening from the surface.

Shaley. Brittle ground.

Sheave. A grooved wheel over which a rope is turned.

Shell-pump. See *Sludger*.

Shelly. Broken ground.

Shift. The duration of day's work—from six to ten hours.

Shoot. To break rock by means of explosives.

Shute. An inclined board-way through which coal is delivered.

Sill. The floor-piece of a timber sett, or that on which the track rests.

Slack. Small dirt or coal.

Slate. Bony coal and hard clay.

Slickenside. The polished surface of the vein, or its walls.

Slitter. See *Pick*.

Slope. An *incline*. It is an *inside slope* when it does not extend to the surface.

Sludger. A cylinder having an upward opening valve at the bottom, which is lowered into a bore-hole to pump out the sludge or fine rock resulting from drillings.

Smift. A slow-burning fuse.

Snore. The hole in the lower part of a sinking- or Cornish-pump through which water enters.

Sollar. The plank flooring of a gallery covering a gutter-way beneath. Also the platform in a shaft between two ladders.

Spears. Pump-rods.

Spilling. A process of timbering through soft ground

Spoon. A slender iron rod with a cup-shaped projection at right angles to the rod, used for scraping drillings out of a bore-hole.

Sprag. A billet of wood used to block the wheels of a car and check its speed. Sprags are permanently used on self-acting inclines. A very steep line requires a sprag in each of

the four wheels, while on a moderate pitch only one may be necessary to block a hind wheel; also, a short prop.

Square-sett. A variety of timbering for large excavations

Squeeze. The closing of a room by the settling of the roof or the rising of the floor. The thinning away of a seam.

Squib. A slow fuse used for igniting an explosive.

Station. An excavation adjoining a shaft for receiving the pump or V balance-bob, or for landing the hoisting conveyances.

Stockwerke. A mass of country rock so impregnated by a congeries of veins as that the whole must be mined together

Stope. A step. The excavation of a vein in a series of steps.

Stoping overhand. Mining a stope upward, the flight of steps being reversed

Stoping underhand. Mining a stope downward in such a series that presents the appearance of a flight of steps

Stowing. The débris of a vein thrown back of a miner to support the roof or hanging wall of an excavation.

Strike. The bearing of a horizontal line through the middle of a vein.

String-rods. A line of surface rods connected rigidly for the transmission of power; used for operating small pumps in adjoining shafts from a central station

Stull. A stick of timber or platform for supporting miners or vein-waste, temporarily or permanently

Stull dirt or stull rock. Material supported upon the stulls.

Stump. A pillar between the gangway and its parallel airway.

Stythe. Carbonic acid gas.

Sump. The lowest point of the workings, usually a prolongation of a shaft, into which the mine water is drained and from which it is pumped.

Swamp. A trough-shaped basin in a coal-mine.

Synclinal. The depression of a seam or stratum.

Tail rope. The secondary rope used for balance, which is attached underneath the cages of a hoisting plant, or at the tail end of the loaded and empty trains of cars on a slope for raising the empty cars or skips.

Tamping. The process of making a bore-hole gas-tight by the use of clay.

Tempering. The act of reheating and properly cooling a bar of metal to any desired degree of elasticity

Throughs or Thirling. A passage cut through a pillar to connect two rooms.

Throw. The amount of dislocation of a vein

Tram. The pair of parallel lines of rails of a trackway.

Trammer. One who pushes cars along the track.

Trap. A door used for cutting off a ventilating current, which is occasionally opened for haulage or passage; guarded by a *trapper*.

Trend. The course of a vein.

Tribute. A system of contract mining by which the miner receives his pay out of the gross value of the ore sold, less a certain deduction for royalty to the mine owner.

Trolley. A small carriage truck having no body. A traveller making connection between two electric wires

Tubbing. An iron or wooden cylindrical lining of shafts.

Tubing. The tube-lining of bore-holes.

Tunnel. A horizontal passage; properly speaking, one with both ends open to the surface; but is applied to one opening at daylight and extending across the country rock to the vein or mine.

Underhand work. Picking or drilling downward.

Underholing. See *Holing* and *Kirving*.

Upcast. An entry through which the air-current rises.

Upraise. An auxiliary shaft, a *mill-hole*, carried from one level up toward another

Vein. A mineral deposit filling a fissure or crevice.

Vug. A cavity in the rock

Wagon breast. One from which ore or coal can be carried by wagon

Wall. The faces of a fissure; the sides of a gallery.

Wall-plate. The long horizontal stick in a shaft timbering frame which is parallel to the vein.

Waste. The débris of an excavation. Gob. Goaf.

Wedging. The material, moss or wood, used to render the shaft-lining tight.

Whim. A hoisting appliance consisting of pulley supporting the hoisting-rope which is wound on a drum turned by a beam attached to a horse

Whip. A hoisting appliance consisting of a pulley supporting the hoisting-rope to which the horse is directly attached.

White-damp. The noxious gas called carbonic-oxide gas

Winch or Windlass. A hoisting machine consisting of a horizontal drum operated by crank-arm and manual labor.

Winning. Recovering or mining.

Winse. A small auxiliary underground shaft sunk from an upper level

Wire-drawing. The operation, accidental or otherwise, of reducing the pressure of steam between the boiler and the cylinder.

Working-barrel. The cylinder in which the pump-piston operates.

Workings. Any underground development from which ore is being extracted.

SIGNALLING.

The writer would suggest the following code of signalling :
The engineer should signal below when he is ready to hoist by raising the bucket or cage a foot or two and lowering again. This is important particularly for the safety of the man who may be engaged in igniting the blasts near to the place of hoist. The strokes should be made at regular intervals. A halt of a few seconds should be made between the signals ; thus :

5 bells-4 bells, would mean send tools to the fourth level.

1 bell : To hoist or to stop if in motion.

2 bells : To lower.

2-1-1 bells : No more hoisting.

2-2-2 bells : To change buckets from ore to water.

3 bells : Man on board ; lower or hoist slow.

4 bells : Stop or start the pump.

2-2 bells : Stop or start the air compressor.

5 bells : Send tools down.

6 bells : Send timbers down.

USEFUL INFORMATION.

The area of a circle is $0.7854 (\text{diameter})^2$.

Ratio of area to circumference is as its radius is to 2.

An acre is 43,560 square feet.

A ton contains 2000 pounds, or $29,166\frac{2}{3}$ Troy ounces.

A Troy pound = 0.822857 avoirdupois pound.

A Troy ounce = 437.5 grains.

An avoirdupois pound = 7000 grains.

A Troy pound = 5760 grains.

A cubic foot of gold = \$300,000.

A cubic foot of silver = \$10,000.

A long ton is 2240 lbs.; a short ton, 2000 lbs.

A bushel of coke = 40 lbs.

A bushel of bituminous coal = 76 lbs. = 2688 cu. in.

1000 feet (board measure) of dry white pine = 4000 lbs.

1000 feet (board measure) of green white pine = 6000 lbs.

One cord of seasoned wood = 128 cu. ft.

A mile of track (rails 16 lbs. per yd.) weighs 25 tons 320 lbs.

A mile of track (rails 25 lbs. per yd.) weighs 39 tons 640 lbs.

A mile of track (rails 35 lbs. per yd.) weighs 55 tons 0 lbs.

A mile of track requires 9 kegs (1780 lbs.) of 3-in. spikes.

A mile of track requires 15 kegs (3110 lbs.) of 4½-in. spikes.

A mile of track requires 20 kegs (3960 lbs.) of 4½-in. spikes.

A mile of track requires 2640 cross-ties (2 ft. apart).

A mile of track requires 528 splice-joints (2 bars, 4 bolts and nuts per joint), each weighing 5 to 10 lbs.

One miner's inch = 2159 cu. ft. per 24 hours = 0.025 cu. ft. per sec.

The pressure of 1 atmosphere is represented by a column of mercury at 32° F., of a height equal to 0.760 metre = 29.92 inches = 14.701 lbs. per sq. in., or by a column of water 33.94 ft. high.

1 lb. air at 32° F., 14.7 lbs. pressure, occupies 12.387 cu. ft.

1 " " 62° F., " " " " 13.141 "

1 " " 180° F., " " " " 16.106 "

1 " " 212° F., " " " " 16.910 "

1 " water at 62° F., " " " " 0.016 "

1 cu. ft. water at 39° F. = 7.49 gallons, weighs 62.425 lbs.

1 " " " 62° F. = " " " 62.355 "

1 gallon " " 62° F. = 231 cubic inches " 8.325 "

1 cu. in. " " 62° F. = 0.00434 gallon, " 0.036 "

1 ton " " 62° F. = 240.3 gals., occupies 35.9 cu. ft.

A col. of water, 1 sq. in. base, 33.947 ft. high = 1 atmosphere.

" " " 1 " " 27.7 in. " = 1 lb. pressure.

" " " 1 " " 1 ft. high = 0.434 lb. press.

" of mercury, 1 " " 1 in. " = 0.4914 "

A bar wro't-iron, 1 " area, 1 ft. long = 3.33 lbs. wt.

" " " 1 in. diam., 1 " " = 2.618 "

TABLE OF WEIGHTS OF VARIOUS SUBSTANCES.

	Weight per Cu. Ft.	Cu. Ft. per Long Ton.		Weight per Cu. Ft.	Cu. Ft. per Long Ton.
Gold.....	1203	1.87	Syenite.....	147 to 184	15.2 to 12.1
Lead.....	710	3.15	Porphyry.....	166 to 171	13.5 to 13.1
Silver.....	625	3.42	Slate.....	162 to 178	13.8 to 12.6
Rolled iron.....	480	4.68	Quartz.....	165.2	13.6
Galena.....	468	4.82	Sandstone.....	130 to 157	17.3 to 14.3
Nickel glance.....	468	4.82	Brick.....	125 to 135	18.1 to 16
Cerussite.....	400	5.60	Clay.....	119.7	18.7
Chalcocite.....	355.7	6.30	Anthracite.....	85.4 to 99	26.2 to 22.6
Magnetite.....	338.6	6.63	Bituminous.....	75 to 83	29.8 to 26.1
Specular iron-ore.....	327.4	6.84	Cannel.....	75	29.8
Pyrites.....	312	7.05	Lignite.....	78 to 84	28.7 to 27
Barytes.....	277.5	8.07	Oak.....	73	30.6
Chalcopyrite.....	262.1	8.55	Ash.....	52.4	42.7
Zinc blende.....	250.0	8.96	White pine.....	44.3	50.5
Hematite.....	250.0	8.96	Yellow pine.....	38.7	57.8
Limestone.....	168.0	13.3	Wood charcoal.....	25 to 39	89.6 to 57.4

EQUIVALENTS OF FRENCH AND ENGLISH MEASURES.

ABBREVIATIONS.—M. = metre; c.m. = centimeter; G. = gramme; L. = litre; ft. = foot; lb. = pound; in. = inch; oz. = ounce; dwt. = pennyweight; gr. = grain; yd. = yard; gal. = gallon; T. = Troy; A. = avoirdupois; sec. = second; sq. = square; cu. = cubic; h. u. = heat unit.

1 M.	= 3.28 ft.	1 ft. per sec.	= 0.305 M. per sec.
1 M.	= 39.39 in.	1 mile per hour	= 0.447 M. "
1 ft.	= 0.3048 M.	1 M. per sec.	= 3.281 ft. "
1 in.	= 0.0254 M.	1000 M. per hour	= 0.621 mile per hour.
1 yd.	= 0.9144 M.	1000 G. per sq. M.	= 0.205 lb. per sq. ft.
1 Gunter's chain	= 20.1168 M.	1000 G. per cu. M.	= 0.0624 lb. per cu. ft.
1 mile	= 1609.35 M.	1 lb. per sq. ft.	= 4883 G. per sq. M.
1 sq. M.	= 1.2 sq. yard.	1 lb. per sq. in.	= 703077.0 G. per sq. M.
1 sq. yd.	= 0.836 sq. M.	1 ton per ft.	= 3.333-333 G. per M.
1 sq. in.	= 0.00065 sq. M.	1 gal. per sq. ft.	= 48.005 L. per sq. M.
1 sq. M.	= 1555.2 sq. in.	1 L. per sq. M.	= 0.0204 gal. per sq. ft.
1 acre	= 4048 sq. M.	1 G. per L.	= 70.116 grs. per gal.
1 cu. in.	= 0.0000164 cu. M.	1 lb. per cu. ft.	= 16020 G. per cu. M.
1 cu. ft.	= 0.02832 cu. M.	1 cu. ft. per lb.	= 0.0624 cu. M. per 1000 G.
1 cu. M.	= 1.35 cu. yd.	1 degree Fahr.	= 0.5555 deg. centigrade.
1 G.	= 15.43 gr.	1 degree Cent.	= 1.8 deg. Fahrenheit.
1 G.	= 0.0022 lb. A.	1 lb. per sq. ft.	= column of mercury 0.00359 M. high.
1 T. gr.	= 0.0648 G.	1 L. of normal air	= 19.955 grains.
1 lb.	= 5760 gr.	1 G. M.	= 0.007233 ft.-lbs.
1 T. lb.	= 373.242 G.	1 ft.-lb.	= 13.82 G. M.
1 A. lb.	= 453.593 G.	772 ft.-lbs.	= 106700 G. M.
1000 G.	= 2 lb., 8 oz., 3 dwt., 0.35 gr. T.	1 calorie	= 3.968 heat units.
1000 G.	= 2 lb., 3 oz., 4 dr., 10.473 gr. A.	1 heat-unit	= 0.252 calorie.
1 fluid oz.	= 0.02957 L.	1 thermal unit	= 0.4536 calorie.
1 quart	= 0.9464 L.	1 h.-u. per lb.	= 0.5555 calorie per 1000 G.
1 gallon	= 3.7854 L.	1 calorie per 1000 G.	= 1 h.-u. per lb.
1 atm. sphere	= 0.760 M.		

TABLE OF HYPERBOLIC LOGARITHMS.

(Base 2.72.)

FOR CALCULATIONS IN THE EXPANSION OF GASES.

Number.	Logarithm.	Number.	Logarithm.	Number.	Logarithm.
1.01	.009	2.10	.741	4.60	1.526
1.02	.019	2.20	.788	4.70	1.547
1.03	.029	2.30	.832	4.80	1.568
1.04	.039	2.40	.875	4.90	1.589
1.05	.048	2.50	.916	5.00	1.609
1.06	.058	2.60	.955	5.10	1.629
1.07	.067	2.70	.993	5.20	1.648
1.08	.076	2.80	1.029	5.30	1.667
1.09	.086	2.90	1.064	5.40	1.686
1.10	.095	3.00	1.098	5.50	1.704
1.11	.104	3.10	1.131	5.60	1.722
1.12	.113	3.20	1.163	5.70	1.740
1.13	.112	3.30	1.193	5.80	1.757
1.14	.131	3.40	1.223	5.90	1.774
1.15	.139	3.50	1.252	6.00	1.791
1.20	.182	3.60	1.280	6.20	1.824
1.25	.223	3.70	1.308	6.40	1.856
1.30	.262	3.80	1.335	6.60	1.887
1.40	.336	3.90	1.360	6.80	1.916
1.50	.405	4.00	1.386	7.00	1.945
1.60	.470	4.10	1.410	7.20	1.974
1.70	.530	4.20	1.435	7.40	2.001
1.80	.587	4.30	1.458	7.60	2.028
1.90	.641	4.40	1.481	7.80	2.054
2.00	.693	4.50	1.504	8.00	2.079

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